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# Inorganic Arsenic Emissions from High-Arsenic Primary Copper Smelters Background Information for Proposed Standards

(U.S.) Environmental Protection Agency
Research Triangle Park, NC

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16. ABSTRACT

Standards of performance to control emissions of inorganic arsenic from new and existing primary copper smelters processing feed materials containing an annual average of 0.7 percent or greater arsenic are being proposed under Section 112 of the Clean Air Act. This document provides information on the background and authority, regulatory alternatives considered, and environmental and economic impacts of the regulatory alternatives.

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# Inorganic Arsenic Emissions from High-Arsenic Primary Copper Smelters - Background Information for Proposed Standards

Emission Standards and Engineering Division

U.S ENVIRONMENTAL PROTECTION AGENCY
Office of Air, Noise, and Radiation
Office of Air Quality Planning and Standards
Research Triangle Park, North Carolina 27711

April 1983

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ENVIRONMENTAL PROTECTION AGENCY

# Background Information and Draft Environmental Impact Statement for Primary Copper Smelters Processing Feed Materials Containing 0.7 Percent or Greater Arsenic

Prepared by:

Jack R Farmer 6/23/83

Jack R. Farmer (Date)
Director, Emission Standards and Engineering Division
U. S. Environmental Protection Agency
Research Triangle Park, NC 27711

1. The proposed standards of performance would limit emissions of inorganic arsenic from existing and new primary copper smelters processing feed materials containing 0.7 percent or greater arsenic. The proposed standards implement Section 112 of the Clean Air Act and are based on the Administrator's determination of June 5, 1980 (45 FR 37886), that inorganic arsenic presents a significant risk to human health as a result of air emissions from one or more stationary source categories, and is therefore a hazardous air pollutant. Only one primary copper smelter, located in the State of Washington, would be affected.

2. Copies of this document have been sent to the following Federal Departments: Labor, Health and Human Services, Defense, Transportation, Agriculture, Commerce, Interior, and Energy; the National Science Foundation; the Council on Environmental Quality; members of the State and Territorial Air Pollution Program Administrators; the Association of Local Air Pollution Control Officials; EPA Regional Administrators; and other interested parties.

3. The comment period for review of this document is 60 days. Mr. Gene W. Smith may be contacted regarding the date of the comment period.

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# 1.0 SUMMARY

## 1.1 STATUTORY AUTHORITY

National emission standards for hazardous air pollutants are established in accordance with Section 112(b)(1)(B) of the Clean Air Act (42 U.S.C. 7412), as amended. Emission standards under Section 112 apply to new and existing sources of a substance that has been listed as a hazardous air pollutant. This study examines inorganic arsenic emission sources at primary copper smelters which process feed material with an annual average inorganic arsenic content of 0.7 percent by weight or greater. This category of primary copper smelters is defined as "high arsenic throughput smelters." The only existing primary copper smelter in the high arsenic throughput smelter category is owned and operated by ASARCO, Incorporated (ASARCO) and located in Tacoma, Washington. No other existing smelters are expected to increase the annual average inorganic arsenic content of their feed materials to or above 0.7 percent and no new smelters are projected to be built during the next 5 years. For this reason, only the ASARCO smelter located in Tacoma, Washington (hereafter referred to as "ASARCO-Tacoma"), is analyzed in this document with respect to the environmental, energy, and economic impacts of regulating the high arsenic throughput smelter category. Primary copper smelters which process feed materials with an annual average arsenic content less than 0.7 weight percent are analyzed in the document, Arsenic Emissions from Low Arsenic Throughput Primary Copper Smelters - Background Information for Proposed Standards (EPA-450/3-83-010a).

## 1.2 REGULATORY ALTERNATIVES

Based on consideration of the control techniques available, three regulatory alternatives were developed for application to arsenic emissions from the ASARCO-Tacoma smelter.

Regulatory Alternative I corresponds to the baseline level of control. No additional controls beyond the controls already in place at the ASARCO-Tacoma smelter to comply with existing regulations (e.g., Washington State implementation plan, OSHA inorganic arsenic worker exposure standard) would be required. Therefore, no national emission standard would be established for arsenic emissions from high arsenic throughput smelters.

Regulatory Alternative II represents control of fugitive arsenic emissions from converter operations at the ASARCO-Tacoma smelter. This alternative is based on capture of the fugitive emissions using a secondary hood with a horizontal air curtain. The captured secondary emission would be vented to a baghouse or equivalent control device for collection.

Regulatory Alternative III would require that arsenic emissions from the ASARCO-Tacoma smelter be reduced to zero. To accomplish this alternative, the ASARCO-Tacoma smelter would be forced to process ores which were virtually free of arsenic content. Implementation of Regulatory Alternative III would therefore result in closure of the ASARCO-Tacoma smelter.

## 1.3 ENVIRONMENTAL IMPACTS

To evaluate environmental impacts, estimates were made of the air quality, water, solid waste, and energy impacts which would result from each of the regulatory alternatives. Table 1-1 summarizes the environmental impact assessments.

Regulatory Alternative I (baseline case) would not change the existing air and nonair quality environmental impacts of operations at the ASARCO-Tacoma smelter. Total arsenic emissions from the smelter would remain at the current level of 282 megagrams (Mg) (311 tons) per year, and there would be no energy or economic impacts attributable to this alternative.

Table 1-1. ENVIRONMENTAL AND ECONOMIC IMPACTS OF REGULATORY ALTERNATIVES

| Regulatory Alternative | Air Impact | Water Impact | Solid Waste Impact | Energy Impact | Noise Impact | Economic Impact |
|---|---|---|---|---|---|---|
| I (no action) | 0 | 0 | 0 | 0 | 0 | 0 |
| II | +4** | 0 | -1** | -1** | 0 | -2** |
| III | +4** | 0 | +1** | +1** | 0 | -4** |

Key:
+ Beneficial Impact
- Adverse Impact

0 No Impact
1 Negligible Impact
2 Small Impact
3 Moderate Impact
4 Large Impact

* Short-Term Impact
** Long-Term Impact
*** Irreversible

Regulatory Alternative II would reduce total arsenic emissions from the ASARCO-Tacoma smelter by 110 Mg (121 tons) per year to a level of 172 Mg (189 tons) per year. The amount of collected particulate matter containing arsenic would be approximately 11 gigagrams (Gg) (12,000 tons) per year. This would increase the amount of solid waste generated at the smelter from 182 to 193 Gg (200,000 to 213,000 tons) per year. This would be about a 6 percent increase. The additional solid waste can be handled by the smelter's existing solid waste disposal system. Since the alternative is based on use of a dry particulate collection device, there would be no direct water pollution impact.

Energy impacts for Regulatory Alternative II would be increased electrical energy consumption. To operate the control system specified by the alternative, annual electrical energy consumption would be 14.6 million kilowatt-hours per year (kWh/yr). Total smelter energy consumption is approximately 2,890 million kWh/yr. Therefore, Regulatory Alternative II would increase the total ASARCO-Tacoma electrical energy consumption by 0.5 percent.

## 1.4 ECONOMIC IMPACT

The economic impact of implementing Regulatory Alternatives II and III were evaluated. Table 1-1 summarizes the assessment of these impacts.

The capital cost for installing the control system specified by Regulatory Alternative II is $3.5 million. This represents a major capital expenditure for ASARCO. However, ASARCO is a major publicly held corporation with a good credit rating and good access to financing. Even considering the possibility of additional capital expenditures for control equipment for the two ASARCO low arsenic throughput smelters, it is determined that the ASARCO-Tacoma smelter would be able to obtain the necessary capital to install the control system at the smelter. The annualized cost to implement Regulatory Alternative II is estimated to be $1.4 million. If ASARCO chooses to absorb the costs by reducing its profit margin, the profitability of the ASARCO-Tacoma smelter could be reduced to 9.6 percent. If ASARCO chooses to maintain its normal profit margin and recover the costs by increasing copper prices, the price of the copper would increase 0.7 to 1.0 percent.

Regulatory Alternative III would reduce total arsenic emissions from the ASARCO-Tacoma smelter by 282 Mg per year to a level of zero emissions. As described above, this total elimination of arsenic emissions could be achieved only through the closure of the ASARCO-Tacoma smelter. Closure of the smelter would eliminate the jobs of approximately 800 ASARCO employees as well as an additional 500 jobs in the local community created by the economic activity of the smelter. This represents approximately 1 percent of the civilian employment in the county where the smelter is located. In addition to providing jobs, ASARCO annually purchases about $20 million worth of goods and services from local companies. ASARCO pays over $2 million per year in State and local taxes. These revenues to local companies and the State and local governments would be eliminated by the closure of the smelter. Finally, the ASARCO-Tacoma smelter is the only facility in the United States producing arsenic trioxide and metallic arsenic for commercial sale. Closure of the smelter would require that all arsenic raw materials for the arsenic chemical manufacturing industry (e.g., pesticides and wood preservative manufacturers) be imported to the United States.

# 2.0 THE PRIMARY COPPER INDUSTRY

## 2.1 GENERAL

Currently, there are 15 primary copper smelters operating or temporarily closed in the United States. Of these, seven are located in Arizona, two in New Mexico, and one each in Nevada, Texas, Utah, Tennessee, Michigan, and Washington. The concentration of copper smelters in the Southwest is due mainly to the local availability of copper-bearing ores. For the 15 copper smelters, smelting capacity totals approximately 1.72 million Mg (1.9 million tons) of smelter product (99 percent "blister" copper) per year.[1] Table 2-1 lists these 15 smelters, their locations, and estimated capacities.

Primary copper production in 1982 was 975,437 Mg (1,075,400 tons).[2] This represented approximately a 60 percent utilization of domestic primary copper smelting capacity. Smelter capacity in the United States appears to be relatively steady, with minimal prospects for substantial additions before 1985. The Bureau of Mines forecasts the total demand for copper in the year 2000 to be between 3.5 and 6.0 teragrams (Tg) ($3.9 \times 10^6$ to $6.6 \times 10^6$ tons). This forecast represents an annual growth of 3.6 percent. However, recycling of scrap and the expansion of hydrometallurgical facilities are expected to accommodate a substantial portion of this increased demand.[3]

### 2.1.1 Raw Materials

Copper ores are generally classified as sulfide, oxide, or native depending on the predominant copper-bearing minerals they contain. Although copper occurs in at least 160 minerals, only a few of these have any commercial importance. Table 2-2 lists the composition of the more important sulfide and oxide minerals from which copper is extracted. Of the primary sulfide minerals, chalcopyrite is the most abundant, followed by bornite and chalcocite. Oxide minerals are produced by the oxidation of primary sulfide minerals under certain

Table 2-1. DOMESTIC PRIMARY COPPER SMELTERS

| Company | Location | Annual capacity[a] | |
|---|---|---|---|
| | | Megagrams | (Tons) |
| ASARCO, Incorporated | El Paso, Texas | 91,000 | (100,000) |
| | Hayden, Arizona | 182,000 | (200,000) |
| | Tacoma, Washington | 91,000 | (100,000) |
| Tennessee Chemical Company | Copperhill, Tennessee | 13,600 | (15,000) |
| Inspiration Consolidated Copper Company | Miami, Arizona | 136,000 | (150,000) |
| Kennecott Copper Corporation | Garfield, Utah | 254,000 | (280,000) |
| | Hayden, Arizona | 71,000 | (78,000) |
| | Hurley, New Mexico | 73,000 | (80,000) |
| | McGill, Nevada | 45,000 | (50,000) |
| Magma Copper Company | San Manuel, Arizona | 181,000 | (200,000) |
| Phelps Dodge Corporation | Ajo, Arizona | 64,000 | (70,000) |
| | Douglas, Arizona | 115,000 | (127,000) |
| | Hidalgo, New Mexico | 163,000 | (179,000) |
| | Morenci, Arizona | 191,000 | (210,000) |
| Copper Range Company | White Pine, Michigan | 52,000 | (57,000) |
| TOTAL | | 1,722,600 | (1,896,000) |

[a] Production of "blister" copper (99 percent Cu).

Table 2-2. MAJOR COPPER-BEARING MINERALS

| Type | Mineral | Formula |
|---|---|---|
| Sulfide | Chalcopyrite | $CuFeS_2$ |
| | Bornite | $Cu_3FeS_3$ |
| | Chalcocite | $Cu_2S$ |
| | Covellite | $CuS$ |
| Oxide | Malachite | $CuCO_3\ Cu(OH)_2$ |
| | Azurite | $2CuCO_3\ Cu(OH)_2$ |
| | Chrysocolla | $CuSiO_3\ 2H_2O$ |
| | Cuprite | $Cu_2O$ |

climatic conditions. When present, these are usually found in the upper portions of copper ore deposits. Native copper consists of almost pure metallic copper. Although found in small amounts in many copper ore deposits, it is essentially unique to the upper peninsula of Michigan.

Copper ores consist of one or more of these copper-bearing minerals disseminated within relatively large quantities of siliceous and other earthy matter. In addition, they also contain varying amounts of other metals including sulfides and oxides of iron, arsenic, antimony, lead, zinc, etc. In the United States, low grade sulfide ores account for 85 to 95 percent of the total primary production.[4] The average tenor of these ores (copper content) is less than 1 percent.[5] Virtually all copper ores processed are beneficiated at the mine. Sulfide ores are crushed, finely ground, and concentrated by froth flotation. Oxide ores are leached with acid, and the dissolved copper is recovered by chemical precipitation on scrap iron. Typically, copper ore concentrates contain about 15 to 30 percent copper.

In addition to copper, metals such as arsenic are also concentrated in the concentration process. The arsenic content of copper concentrates processed at U.S. copper smelters is highly variable, ranging from a few parts per million to several percent.

Table 2-3 presents the amount of arsenic input to the domestic smelters, based on information received from the smelters in early 1983.[6,7,8,9,10,11,12] The ASARCO-Tacoma smelter is a custom smelter, and is the only U.S. smelter which recovers arsenic trioxide as a by-product. Therefore, the smelter processes ore materials which typically contain high concentrations of arsenic, obtained from various domestic and foreign sources. As shown in the table, the ASARCO-Tacoma smelter introduces more arsenic in the feed material to the smelter than the combined total from all other smelters. The smelter also handles substantial quantities of lead plant by-products and flue dust from other copper smelters. As a result, more arsenic is potentially released from this smelter as process emissions and fugitive emissions than from the other smelters combined.

### 2.1.2 Process Description[13]

The pyrometallurgical process used for the extraction of copper from sulfide ore concentrates is based on iron's strong affinity for oxygen as compared to copper's weak affinity for oxygen. The purpose of smelting is to separate the copper from the iron, sulfur, and gangue materials. Conventional practice includes three operations:

1. Roasting (optional) to remove a portion of the concentrate sulfur content.
2. Smelting of roasted calcines or unroasted ore concentrates and fluxes in a furnace to form slag and copper-bearing matte.
3. Converting (oxidizing) of the matte in a converter to form blister copper (about 99 percent pure copper).

Figure 2-1 presents a pictorial representation of the copper smelting process. Figure 2-2 illustrates the three basic operations employed as well as materials entering or leaving each operation. Briefly, the smelting of copper concentrates and precipitates is accomplished by melting the charge and suitable fluxes in a smelting furnace. Part or all of the concentrates may receive a partial roast to eliminate some of the sulfur and impurities such as arsenic. In the smelting furnace, the lighter impurities combine and float to the top as slag to be skimmed off and discarded, while the copper, iron, most of the sulfur, and any contained precious metals form a product known as matte which collects in and is drawn off from the lower part

## Table 2-3. ARSENIC INPUT IN THE FEED TO DOMESTIC COPPER SMELTERS

| Plant | Arsenic content of feed[a,b] | | |
|---|---|---|---|
| | Percent | kg/hr | (lb/hr) |
| ASARCO-Tacoma | 4.0 | 991[c] | (2185) |
| ASARCO-Hayden | 0.6 | 170 | (375) |
| ASARCO-El Paso | 0.5 | 142[d] | (314) |
| Kennecott-Garfield | 0.14 | 118[e] | (261) |
| Kennecott-McGill | 0.4 | 81[f] | (179.3) |
| Phelps Dodge-Ajo | 0.3 | 47 | (103) |
| Inspiration-Miami | 0.033 | 18[g] | (41.1) |
| Phelps Dodge-Hidalgo | 0.018 | 14 | (30.6) |
| Phelps Dodge-Douglas | 0.03 | 11 | (24) |
| Kennecott-Hayden | 0.015 | 8.0 | (17.7) |
| Phelps Dodge-Morenci | 0.006 | 4.5[h] | (9.99) |
| Magma-San Manuel | 0.006 | 2.0 | (4.39) |
| Tennessee Chemical Company-Copperhill | 0.0004 | 1.3 | (2.9) |
| Kennecott-Hurley | 0.0005 | 1.0 | (2.14) |
| Copper Range-White Pine | 0.008 | 0.7[i] | (1.53) |

[a] The feed is a mixture of concentrates, precipitates, lead smelter by-products, and smelter reverts.

[b] Does not include recycled flue dusts and other intermediates, except where noted.

[c] 50 kg/hr (111 lb/hr) of this amount is fed directly to the arsenic plant.

[d] 51 kg/hr (112 lb/hr) of this amount is fed directly to the converters.

[e] 3.5 kg/hr (7.8 lb/hr) of this amount is fed directly to the converters.

[f] 6.5 kg/hr (14.2 lb/hr) of this amount is fed directly to the converters.

[g] 0.3 kg/hr (0.6 lb/hr) of this amount is fed directly to the converters.

[h] 0.2 kg/hr (0.35 lb/hr) of this amount is fed directly to the converters.

[i] 0.1 kg/hr (0.22 lb/hr) of this amount is fed directly to the converters.

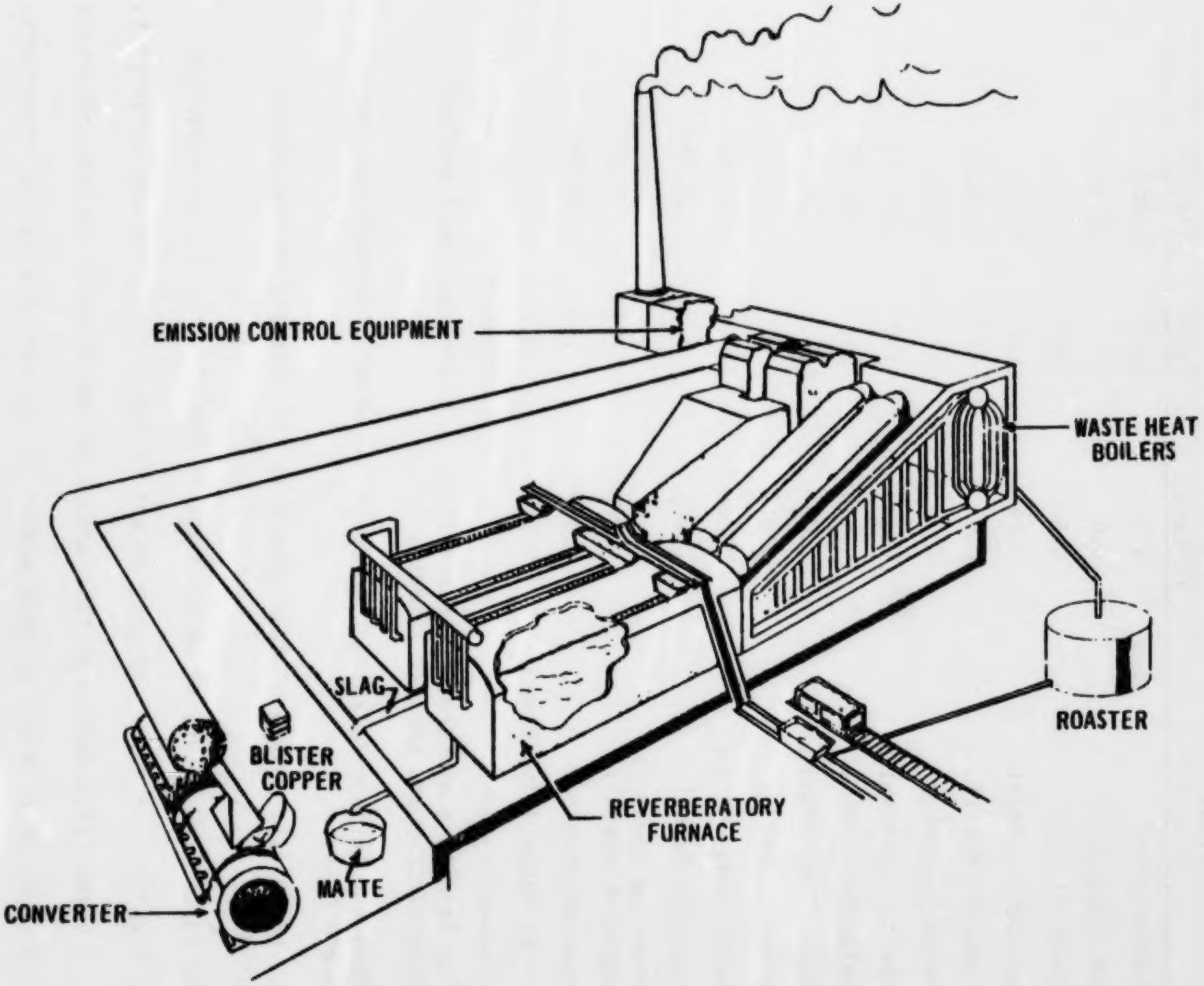

EMISSION CONTROL EQUIPMENT
WASTE HEAT BOILERS
SLAG
BLISTER COPPER
ROASTER
REVERBERATORY FURNACE
CONVERTER
MATTE

Figure 2-1. Primary Copper Smelter

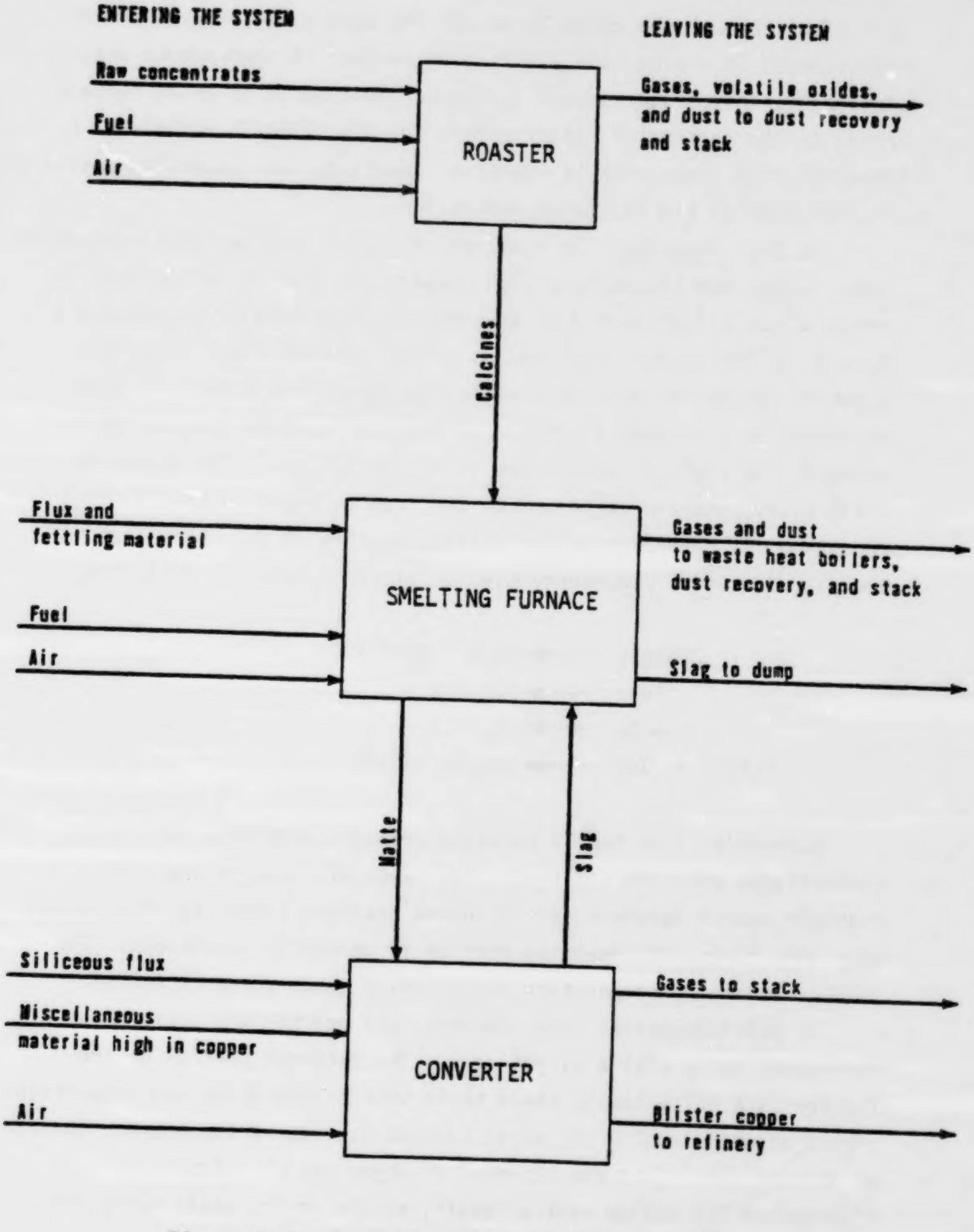

ENTERING THE SYSTEM
LEAVING THE SYSTEM
Raw concentrates
Fuel
Air
ROASTER
Gases, volatile oxides, and dust to dust recovery and stack
Calcines
Flux and fettling material
Fuel
Air
SMELTING FURNACE
Gases and dust to waste heat boilers, dust recovery, and stack
Slag to dump
Matte
Slag
Siliceous flux
Miscellaneous material high in copper
Air
CONVERTER
Gases to stack
Blister copper to refinery

Figure 2-2. Primary Copper Smelting Process

of the furnace. The molten matte is transferred to a converter where air blown through the matte burns off the sulfur, oxidizes the iron for removal in a slag, and yields a 99 percent blister copper product. Typically, the blister copper is further refined in an anode furnace prior to the casting of copper anodes for electrolytic refining. A more detailed discussion of roasting, smelting, and converting operations is presented in the following subsections.

2.1.2.1 Roasting. In roasting of copper sulfide ore concentrates, concentrates are heated to a high temperature (but below the melting point of the constituents) in an oxidizing atmosphere to eliminate a portion of the sulfur contained as sulfur dioxide ($SO_2$); to remove volatile impurities such as arsenic, antimony, and bismuth; and to preferentially convert a portion of the iron sulfides present to iron oxides. The roasted concentrate is called calcine. The degree of roast (i.e., the amount of sulfur and iron oxidized in the roasting operation) is dependent on the desired quality of the charge to the smelting furnace. Representative reactions include the following:

$$2CuFeS_2 \longrightarrow Cu_2S + 2FeS + S$$

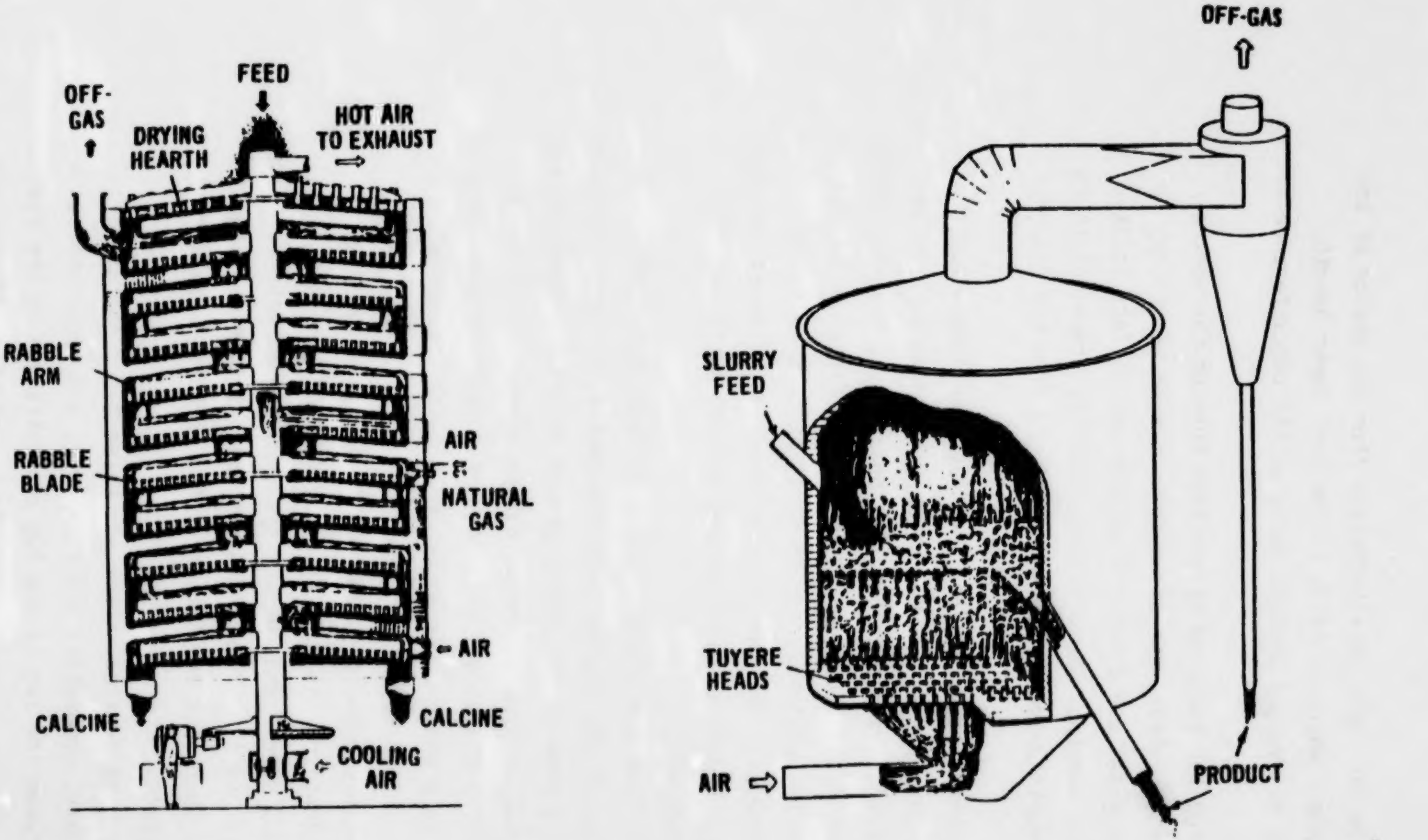
$$FeS_2 \longrightarrow FeS + S$$

$$S + O_2 \longrightarrow SO_2$$

$$4FeS + 7O_2 \longrightarrow 2Fe_2O_3 + 4SO_2$$

Currently, 7 of the 15 existing primary copper smelters roast concentrates prior to smelting. Two types of roasters are used: multiple-hearth roasters and fluid-bed roasters (refer to Figure 2-3). With both types, the roasting process is generally autogenous. The roaster operating temperature is typically about 650°C (1,200°F).

In multiple-hearth type roasters, the hearths are constructed of refractory brick with a slight arch. The external portion of the furnace is a brick-lined, steel shell with hinged doors and inspection plates at each level. The moist concentrate enters the roaster through an annular opening to the top-most or dryer hearth. Rabble arms, attached to the hollow central shaft, rotate as the shaft turns and plow through the charge to continuously expose fresh surfaces to the oxidizing air. The rabble blades are set at an angle and, in addition


OFF-GAS
FEED
HOT AIR TO EXHAUST
DRYING HEARTH
RABBLE ARM
RABBLE BLADE
AIR
NATURAL GAS
AIR
CALCINE
CALCINE
COOLING AIR
OFF-GAS
SLURRY FEED
TUYERE HEADS
AIR
PRODUCT

MULTI-HEARTH ROASTER

FLUID-BED ROASTER

Figure 2-3. Calcine Roaster

to stirring the material, move it alternately from the center of the hearth to the periphery where it falls to the next lower hearth. Finished calcine is discharged through holes on the circumference of the bottom hearth.

The air required for roasting is admitted through the central shaft and, by means of valves, the air supply to each hearth may be regulated. Roaster gases are drawn off through gas outlets located just below the dryer hearth. The discharge from the dryer hearth to the top roasting hearth prevents the escape of gas from the interior of the roaster.

Fluid-bed roasters are cylindrical, refractory-lined vessels equipped with diffusion plates in the bottom containing tuyeres or bubble caps through which air is blown from the bottom. Finely ground material (60 percent minus 200 mesh) is introduced either as a slurry through a feed pipe or relatively dry (6 to 12 percent moisture) through a screw conveyor. The feed is continuously delivered into the combustion chamber. Roasting occurs as the sulfide particles fall through the oxidizing air.

Combustion air from the windbox passes through distribution plates at the bottom of the chamber into the combustion zone. Because of the large surface area of the finely ground material exposed to the airstream, the residence time in the oxidizing atmosphere is short. The reaction is self-sustaining. Oil, gas, or pulverized coal burners are required only to preheat the roaster to combustion temperature.

Roaster gases are drawn off through flues at the top of the chamber and immediately pass to cyclone collectors, followed by cooling and final dust collection. As much as 85 percent of the feed is carried with the gas stream and, hence, cyclones are an integral part of the roasting operation. Screw conveyors collect this finished calcine as well as that from the bottom of the combustion chamber for discharge into hoppers.

For either type of roaster, there are three major operating variables: feed rate, combustion air flow rate, and temperature. A key difference between the two is the $SO_2$ concentration in the roaster offgases. Sulfur dioxide concentrations in the roaster offgases are considerably higher for fluid-bed roasters than for multi-hearth roasters due to the lower total air volume. Average stack gas $SO_2$ concentration is about 12 percent with a maximum of 18 percent for fluid-bed roasters and only 3 to 6 percent for multi-hearth roasters.[14]

2.1.2.2 Smelting. Smelting is the pyrometallurgical process in which solid material is melted and subjected to certain chemical changes. During copper smelting, hot calcines from the roaster or raw, unroasted concentrates are melted in a smelting furnace with siliceous or limestone flux. Converter slag, collected dust, oxide ores, and any other material rich in copper may be added to the furnace charge. Copper and iron which are present in the charge combine with sulfur to form a stable cuprous sulfide. Excess sulfur unites with iron to form a stable ferrous sulfide (FeS). The combination of the two sulfides, known as matte, collects in the lower area of the furnace and is removed. Such mattes may contain from 15 to 50 percent copper, with a 40 to 45 percent copper content being most common. Mattes may also contain impurities such as sulfur, antimony, arsenic, iron, and precious metals.

The remainder of the molten mass containing most of the other impurities is known as slag. Slag is of lower specific gravity, floats on top of the matte, and is drawn off and discarded. Slags in copper smelting are ideally represented by the composition $2FeO \cdot SiO_2$, but contain alumina from the various charge materials and calcium oxide which is added for fluidity. Since slags are discarded, the copper contained in the slag is a major source of copper loss in pyrometallurgical practice. Copper concentration in the slag increases with increasing matte grade, which limits the matte grades normally obtained in conventional practice to below 50 percent copper.

Currently, conventional reverberatory furnaces are used at 11 of the 15 existing primary copper smelters. Two smelters employ electric furnaces, while one smelter employs the Outokumpu flash furnace and another a Noranda continuous smelter. Two smelters will be modifying one or more of their reverberatory furnaces to convert them to oxygen-sprinkle smelting, while two more are planning to retire their reverberatory furnaces within the next few years and install Inco flash smelting furnaces. In a reverberatory furnace (Figure 2-4), fossil

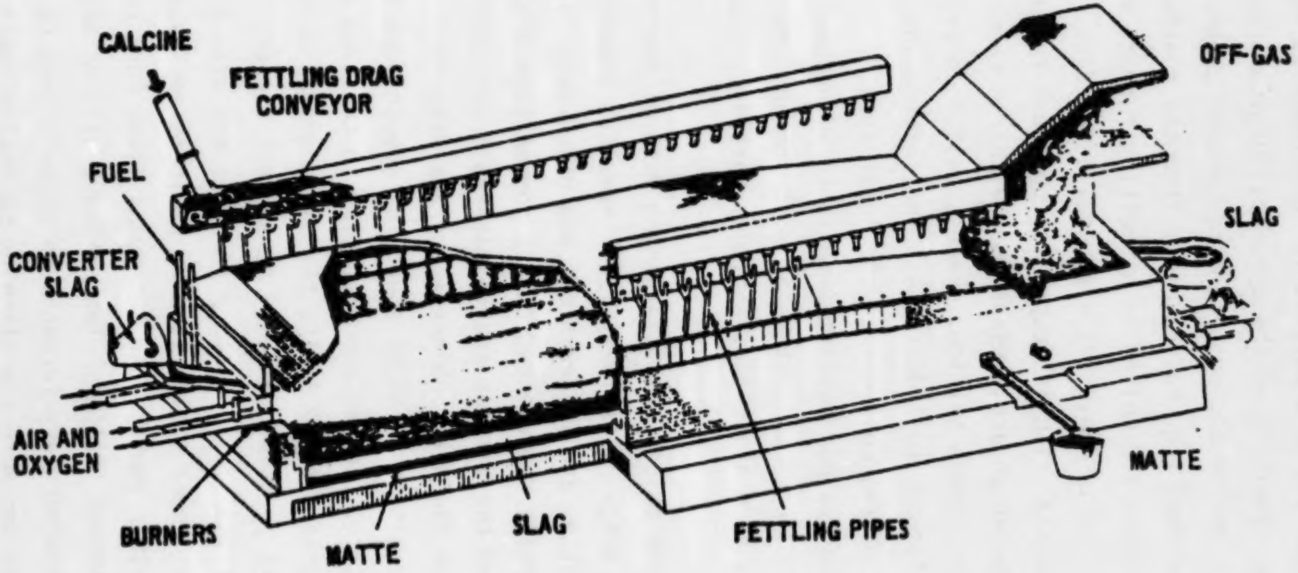

CALCINE
FETTLING DRAG CONVEYOR
FUEL
CONVERTER SLAG
AIR AND OXYGEN
BURNERS
MATTE
SLAG
FETTLING PIPES
MATTE
SLAG
OFF-GAS

Figure 2-4. Reverberatory Smelting Furnace

fuels such as oil or natural gas are burned above the copper concentrates being smelted. The furnace is a long, rectangular structure, generally about 11 m (36 ft) in width and 40 m (131 ft) in length, with an arched roof and burners at one end. Flames from the burners may extend half the length of the furnace. Temperatures at the firing end of the furnace exceed 1,500°C (2,730°F). Part of the heat in the combustion gas radiates directly to the charge lying on the hearth below, while a substantial part radiates to the furnace roofs and walls and is reflected down to the charge. The roofs of the older reverberatory furnaces are sprung-arch silica roofs, while almost all newer furnaces have suspended roofs of basic refractory.

Over the years, two types of reverberatory furnaces have evolved, each with its own specific charging methods. The older type is the deep bath reverberatory furnace which contains a large quantity of molten slag and matte at all times. In modern, deep bath reverberatory furnaces, the molten material is held in a refractory crucible with cooling water jackets along the sides, which greatly diminishes the danger of a breakout of the liquid material. In deep bath smelting, several methods exist for charging. Wet concentrates can be charged using slinger belts (high-speed conveyors) that spread the concentrates on the surface of the molten bath. Dry concentrates or calcines from the roaster can be charged through the roof or via a Wagstaff gun (an inclined tube). Roof charging (side charging) is rarely practiced in conjunction with deep bath smelting because of dusting problems with fine dry calcine and explosion potential with green charge. Wagstaff guns minimize these problems and are commonly used.

Side or roof charging is usually used with green charge. With the charge dropped through a series of feed holes in the arch near the walls, a buildup of material forming banks results. The banks slowly melt and serve as protection to the side walls, eliminating the requirement for special cooling.

Combustion gases contain from 15 to 45 percent of the sulfur in the original charge depending primarily upon whether or not the concentrate was roasted. However, because of the high volume of combustion air, $SO_2$ concentrations are low, with averages varying from fluid-bed roasters, converters, and other types of smelting furnaces, cannot be economically utilized as feed for sulfuric acid plants.

Electric smelting furnaces provide the heat necessary for smelting copper ore concentrates by allowing carbon electrodes to come into contact with the molten bath within the furnace. The electrodes dip into the slag layer of the bath, forming an electrical circuit. When an electric current is passed through this circuit, the slag resists its passage, generating heat and producing smelting temperatures. Charge concentrates and fluxing materials are fed through the roof, and a layer of unsmelted charge covers the molten bath. Heat is transferred from the hot slag to the charge floating on its surface, and as the copper concentrates and fluxes are smelted, they settle into the bath forming slag and matte. The chemical and physical changes occurring in the molten bath are similar to those occurring in the molten bath of a reverberatory furnace.

In flash smelting, copper sulfide ore concentrates are smelted by burning a portion of the iron and sulfur contained in the concentrates while they are suspended in an oxidizing environment. As such, the process is quite similar to the combustion of pulverized coal. The concentrates and fluxes are injected with preheated air, oxygen-enriched air, or even pure oxygen, into a furnace of special design. Then, smelting temperatures are attained as a result of the heat released by the rapid, flash combustion of iron and sulfur. Flash smelting technology has been developed by two companies: International Nickel Company (Inco) in Canada and Outokumpu Oy in Finland. The major difference between the two technologies is in the design of the smelting furnace and the oxidizing environment within the furnace. The Inco furnace uses pure oxygen, while the Outokumpu furnace employs preheated air or oxygen-enriched air as the oxidizing medium.

Two smelters are currently planning to convert one or more of their reverberatory furnaces to oxygen-sprinkle smelting. In essence, this conversion will allow the existing reverberatory furnaces to behave like flash furnaces. Specially designed burners positioned on the furnace roof are used to introduce and disperse a mixture of primarily dried concentrates and oxygen. The heat required for smelting is generated from the flash combustion of the sulfur in the mixture.

The result is an increase in furnace efficiency and the production of a strong $SO_2$ process gas stream suitable for treatment in a sulfuric acid plant.

2.1.2.3 Converting. Matte produced in the reverberatory furnace is transferred in ladles to the converters using overhead cranes. Fourteen of the 15 smelters use converters of the cylindrical Pierce-Smith type, the most common size being 4 by 9 m (13 by 30 ft). Figure 2-5 is a sketch of a Pierce-Smith copper converter. An alternative to the Pierce-Smith converter is the newer Hoboken or "siphon" converter. The Hoboken converter, currently used by one of the domestic smelters, is essentially the same as the Pierce-Smith converter except that it is fitted with a side flue located at one end of the converter and shaped as an inverted "U." This flue arrangement permits siphoning of the converter gases from the interior of the converter, using variable-speed fans and dampers, directly to the offgas collection system. By maintaining a slightly negative pressure at the converter mouth, it is possible to minimize or eliminate emissions. Problems in improper draft at the converter mouth have been reported by Inspiration Consolidated Copper Company, the only domestic user of Hoboken converters. One of the five Hoboken converters operated by Inspiration has been modified to eliminate the siphon area. The modification has resulted in improved performance, and Inspiration intends to modify its four remaining smelters in similar fashion.[16]

In both Pierce-Smith and Hoboken converters, air is blown from the side through a series of openings called tuyeres. During the initial blowing period (the slag blow), FeS in the matte is preferentially oxidized to FeO and $Fe_3O_4$, and sulfur is removed with the offgases as $SO_2$. Flux is added to the converter to combine with iron oxide and forms a fluid iron silicate slag. When all the iron is oxidized, the slag is skimmed and poured off from the furnace at various times during slag formation, leaving behind "white metal" or molten $Cu_2S$.

During this stage, fresh matte is charged into the converter, and the slag blowing continues until a sufficient quantity of white metal has accumulated. When this happens, the white metal is oxidized with air to blister copper during the "copper blow." The blister copper is removed from the converter and cast or subjected to additional fire

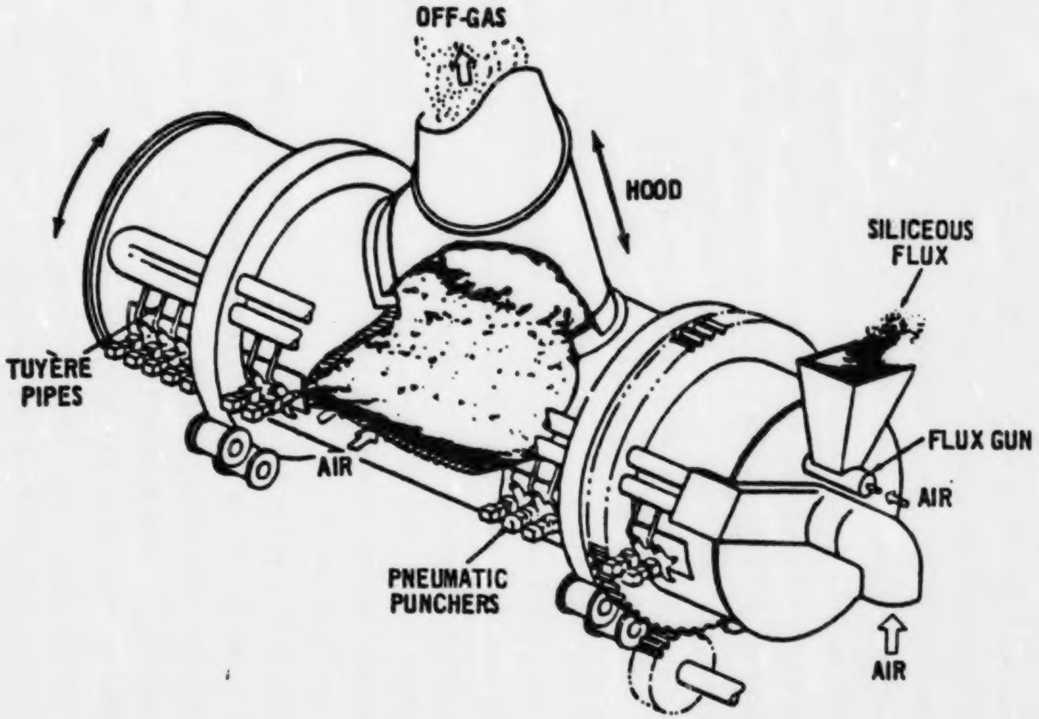

OFF-GAS
HOOD
SILICEOUS FLUX
TUYÈRE PIPES
FLUX GUN
AIR
AIR
PNEUMATIC PUNCHERS
AIR

Figure 2-5. Copper Converter

refining prior to casting. Converter blowing rates can vary between 340 and 855 normal $m^3$/min (12,000 to 30,000 scfm) of air.

In general practice, the matte is added to the converter in two to six steps, each step followed by oxidation of much of the FeS from the charge. The resulting slag is poured from the converter after each oxidation step, and a new matte addition is made. In this way, the amount of copper (as matte) in the converter gradually increases until there is a sufficient amount for a final copper-making "blow." At this point, the FeS in the matte is blown down to about 1 percent, a final slag is removed, and the resulting white metal is oxidized to blister copper. The converting process is terminated when copper oxide begins to appear with the liquid copper. The offgas flow rate leaving the primary head of converters typically ranges from 850 to 1,260 $Nm^3$/min (30,000 to 45,000 scfm). The average $SO_2$ concentration in these gases is normally in the range of 4 to 5 percent during the slag blow and 7 to 8 percent during the copper blow. Values of the overall average $SO_2$ concentration in the offgases (after gas cleaning) from existing domestic converting operations fall in the range of 1.6 to 6.5 percent on a dry basis. The industry-wide average value, as weighted by the respective plant flow rates, is 4.6 percent.[17]

2.1.2.4 Refining. Virtually all copper produced by matte smelting is subsequently electrorefined. For this reason, the final liquid copper product of the smelter must be suitable for the casting of strong, thin anodes. In a majority of cases, anode copper is fire refined directly from molten blister copper.

Fire refining is performed in rotary-type refining furnaces resembling Pierce-Smith converters, or in small hearth furnaces. The rotary-type predominates when molten blister copper is treated directly, while hearth furnaces are used when melting of solid charges is practiced. The temperature of operation is about 1,130 to 1,150°C, which provides sufficient superheat for the subsequent casting of anodes. There is very little heat produced by refining reactions, and some combustion of fuel is necessary to maintain the temperature in the furnace.

Dimensions of the rotary-type refining furnaces vary; however, a 4 by 9 m (13 by 30 ft) furnace may be regarded as typical. Gas flow rates are generally low so as to accurately control the metal composition. Gas pressures at the tuyeres are near 250 to 600 kPa (2.5 to 6 atm). Refining a 250-ton charge of copper requires 3 to 5 hours: 1/2 to 1 hour for the oxidation step, and the remainder for the deoxidation ("poling") step.

A typical sequence of events in rotary anode furnace refining is:

1. Molten blister copper is added to the anode furnace as it becomes available from the converters until about 150 to 300 Mg (165 to 331 tons) have been accumulated;

2. The accumulated charge is then oxidized by blowing air through the tuyeres until the sulfur content is lowered to 0.001 to 0.003 percent S, at which time a small ingot sample of copper shows a slight contraction or small hole; and at which time the oxygen level in the copper is about 0.6 percent;

3. The oxygen is then removed from the copper by blowing natural gas, reformed natural gas, or propane through the tuyeres. The oxygen level in the anode copper is 0.005 to 0.2 percent after this operation, which gives a "flat set" to the anodes when they are cast. The correct "end point" is determined by casting a sample of the copper (which should set to a flat surface) or by continuously analyzing for oxygen with an oxygen probe;

4. Finally, the liquid metal may be covered with low sulfur coke to prevent reoxidation of the copper.

Several older modifications of this process are still being used in which the air is introduced via steel lances, and in which the oxygen-removal step is performed by lowering large logs or poles of green wood into the copper (thereby providing the necessary hydrocarbons). Both of these steps are clumsy, and are being discontinued.

Similarly, the use of the hearth type of anode furnace has by and large been discontinued except where scrap (including anode scrap) and/or blister copper are melted. A typical anode hearth furnace resembles a small reverberatory furnace o width 5 m (16 ft), length 15 m (49 ft), height 3 m (10 ft), inside dimensions o capable of holding 300 Mg (331 tons) of copper. In hearth furnaces, the air and hydrocarbons are introduced into the copper by submerging steel lances into the molten bath. Wooden poles are often used for the oxygen removal step.

### 2.1.3 Description of the ASARCO-Tacoma Smelter

The ASARCO smelter at Tacoma, Washington, is a custom smelter which processes copper ore concentrates, precipitates, and smelter by-products from numerous domestic and foreign sources. The smelter produces about 320 Mg (353 tons) of anode copper per day at full smelt and houses the only arsenic production facility in the United States. Copper smelter facilities include 10 roasters, (6 Herreschoff multi-hearth roasters and 4 C & W multi-hearth roasters), 2 reverberatory smelting furnaces, 3 Pierce-Smith converters (a fourth Pierce-Smith converter is used as a holding furnace only), and 2 tilting anode furnaces. Arsenic production facilities consist of three Godfrey roasters, three arsenic trioxide settling chambers or kitchens, storage facilities, and a metallic arsenic plant.

The roaster charge, which consists of a blend of concentrates, precipitates, lead speiss, flue dust, and fluxing materials, typically contains 3 to 4 percent arsenic and 7 to 10 percent moisture. At full smelt, six or seven roasters are used. Charging is continuous. The calcine produced, about 45 Mg (50 tons) per hour, is intermittently discharged from hoppers located below the roasters into larry cars for transport to one of two reverberatory furnaces. Typically, two 5.9-Mg (6.5-ton) cars are charged every 15 minutes.

The smelter has two reverberatory smelting furnaces, each with an approximate smelting capacity of 1,090 Mg (1,200 tons) per day. The designated Number 2 furnace is used almost exclusively. The furnace measures 33.5 m (110 ft) in length and 9.8 m (32 ft) in width and is fired by either oil or natural gas. Furnace charging is accomplished by discharging the larry cars through one of four Wagstaff guns located along the furnace sidewalls. Typically, it takes less than 1 minute to discharge one car. At full smelt, two cars are discharged about every 15 minutes. To minimize potential fugitive emissions during charging, a manual control override is used which simultaneously opens the furnace flue control damper and reduces the fuel supply to the furnace fire prior to each charge to prevent pressure surges in the furnace.

Matte is tapped from the furnace as required by converter operations through one of four tapping ports. Only one tapping port is used at a time. The matte flows through a cast copper launder to a 4.25 $m^3$ (150 $ft^3$) cast steel ladle for transfer to the converters. At full smelt, about 45 ladles are transferred per day. It takes about 7 minutes to fill one ladle. Similarly, slag is skimmed through one of two tapping ports as required to maintain the proper slag level in the furnace and transferred to a 5-pot slag train for transit to the slag dump. Once tapped, the slag flows through a cast steel launder and into a 2.8 $m^3$ (100 $ft^3$) cast steel slag pot. At full smelt, about 20 5-pot slag trains are dumped per day. Each train takes about 15 minutes to fill.

Matte from the reverberatory furnace is transferred by crane to one of the Pierce-Smith converters. The converters measure 4.0 m (13 ft) in diameter by 9.1 m (30 ft) in length. In addition to copper matte, smelter reverts and cold dope materials are also processed. Typically, only two converters are on blow at any one time. A converter cycle normally takes from 10 to 12 hours. With dilution air, the offgas flow per blowing converter is about 1,130 normal $m^3$/min (40,000 scfm) and contains from 3 to 4 percent $SO_2$. Blister copper produced is transferred to the anode furnaces for refining and casting. The slag skimmed from the molten charge is recycled to the reverberatory furnace.

Arsenic-laden dust recovered from the particulate control systems servicing the multi-hearth roasters, the reverberatory furnace, and the converters is transferred to the arsenic recovery plant. The dust is charged to one of the Godfrey roasters where the arsenic trioxide contained in the dust is volatilized. The roaster calcine or troughs, now low in arsenic content, are discharged to open freight cars and subsequently recycled through the copper smelter complex to resorb the copper value. The volatilized arsenic is passed through a series of arsenic settling chambers, called kitchens, where the temperature of the gas and vapor are controlled and the arsenic trioxide is condensed. The kitchens are rectangular structures containing multiple baffled chambers. There are 3 arsenic kitchens, 1 with 10 chambers and 2 with 15 chambers. The arsenic-laden gas enters the first chamber at approximately 205°C (400°F) and exits the final chamber at 100°C (212°F) or less. The exhaust gases from the kitchens are treated in a baghouse prior to being discharged to the atmosphere. The condensed arsenic trioxide

(90 to 95 percent pure) is removed periodically from the settling kitchens and placed in storage. This material is shipped as a product.

ASARCO-Tacoma also operates an elemental arsenic plant, independent of the copper circuit, in which purchased, refined arsenic trioxide is reduced to metallic arsenic. Offgases from the two metallic arsenic furnaces pass through condensers (one per furnace) in which the gaseous arsenic is collected as a solid and periodically removed. The metallic arsenic is loaded into barrels for shipment. Offgases from the arsenic condensers are treated in the arsenic trioxide plant baghouse.

## 2.2 ARSENIC DISTRIBUTION AND EMISSIONS AT ASARCO-TACOMA

Arsenic elimination during copper smelting occurs primarily by volatilization and slagging. The bulk of the arsenic is volatilized in most pyrometallurgical processes and removed with the offgases. In the presence of oxygen, elemental arsenic or arsenic sulfides oxidize to arsenic trioxide, which is extremely volatile. However, in an oxidizing atmosphere, the arsenic trioxide ($As_2O_3$) may oxidize to the higher oxide ($As_2O_5$) which is less volatile and forms stable nonvolatile arsenates with other metallic oxides.

The elimination of arsenic in the smelting furnace by slagging is dependent upon the concentration or partial pressure of the arsenic trioxide.[18] If the partial pressure of the oxide is sufficient to allow it to penetrate the slag layer and thus escape, volatilization will occur. However, if the partial pressure is not great enough to allow passage through the slag layer, the arsenic will be retained in the slag and thus be eliminated on slag disposal. The magnitude of the partial pressure is dependent upon the amount of arsenic present in the process stream.

Arsenic emissions from primary copper smelters can be categorized as process and fugitive emissions. Process emission sources include primary offgas emissions from roasting, smelting, and converting operations. Fugitive emissions include those emissions which escape from process flow streams due to leakage through process exhaust systems, material transfer systems, or ineffective capture at the source of generation.

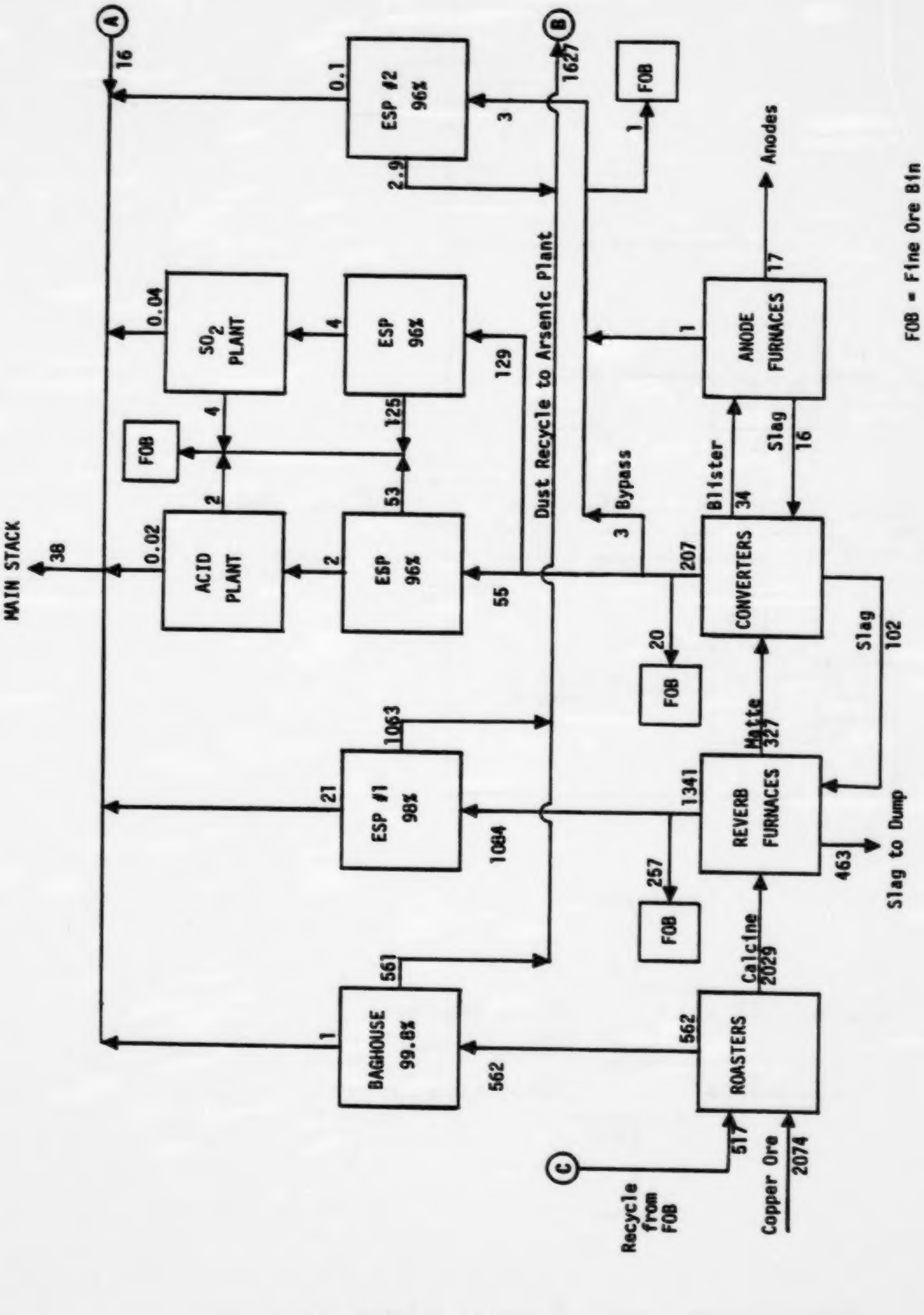
### 2.2.1 Arsenic Distribution and Process Emissions at ASARCO-Tacoma

The arsenic distribution for this smelter is shown in Figure 2-6. The arsenic input of 991 kg/hr (2,185 lb/hr) includes the arsenic found in the concentrates, precipitates, and lead smelter by-products and reverts, as well as the input to the metallic arsenic plant. Figure 2-6 is based on a detailed arsenic material balance supplied by ASARCO.[9] Process emissions are calculated based on the control systems currently in place at ASARCO-Tacoma. Fugitive emissions are not included in the material balance. Arsenic flow rates are based on the total tonnages transferred in a year (1982) divided by 8,544 hours of operation. These rates, therefore, do not reflect instantaneous flow at full capacity. It should also be noted that operations curtailment at the ASARCO-Tacoma smelter in 1982 was approximately 25 to 30 percent.[19]

The arsenic collection efficiency of the particulate control equipment is significantly affected by the temperature at which arsenic trioxide enters and leaves the equipment. For this reason, all assigned control efficiencies are dependent on the temperature at which the efficiencies were determined. In the arsenic material balance shown in Figure 2-6, estimated arsenic collection efficiencies for control devices were based on information presented in Section 3.2.1. Based on the results of emission source testing at the smelter, the roaster baghouse was assigned an efficiency of 99.8 percent and electrostatic precipitator No. 1 was assigned an efficiency of 98 percent. Electrostatic precipitator No. 2 and the electrostatic precipitators upstream of the acid and sulfur dioxide plants were assumed to be 96 percent efficient based on engineering judgment. The chemical plants themselves were assigned an arsenic removal efficiency of 99 percent. An efficiency rating of 98 percent was assumed for the new arsenic plant baghouse.

Table 2-4 presents a summary of process arsenic emissions estimated for ASARCO-Tacoma. These estimates are based on inputs to the various process steps and collection efficiencies of associated control equipment.

In the absence of any controls, the major sources of arsenic process emissions at ASARCO-Tacoma in order of magnitude of emissions are the reverberatory furnaces, the arsenic plant, the multi-hearth


MAIN STACK
38
A
16
C
B
1627
Recycle from FOB
517
Copper Ore
2074
ROASTERS
562
Calcine
2029
BAGHOUSE 99.8%
562
1
561
REVERB FURNACES
257
FOB
1341
463
Slag to Dump
Matte
327
ESP #1 98%
1084
21
1063
CONVERTERS
20
FOB
207
Slag
102
Blister
34
Slag
16
Bypass
3
ESP 96%
55
2
53
ACID PLANT
0.02
2
FOB
4
125
ESP 96%
129
4
SO2 PLANT
0.04
Dust Recycle to Arsenic Plant
ANODE FURNACES
1
17
Anodes
ESP #2 96%
2.9
0.1
3
1
FOB

FOB = Fine Ore Bin

*Arsenic material flow expressed in units of lb/hr

Figure 2-6. ASARCO-Tacoma Arsenic Material Balance

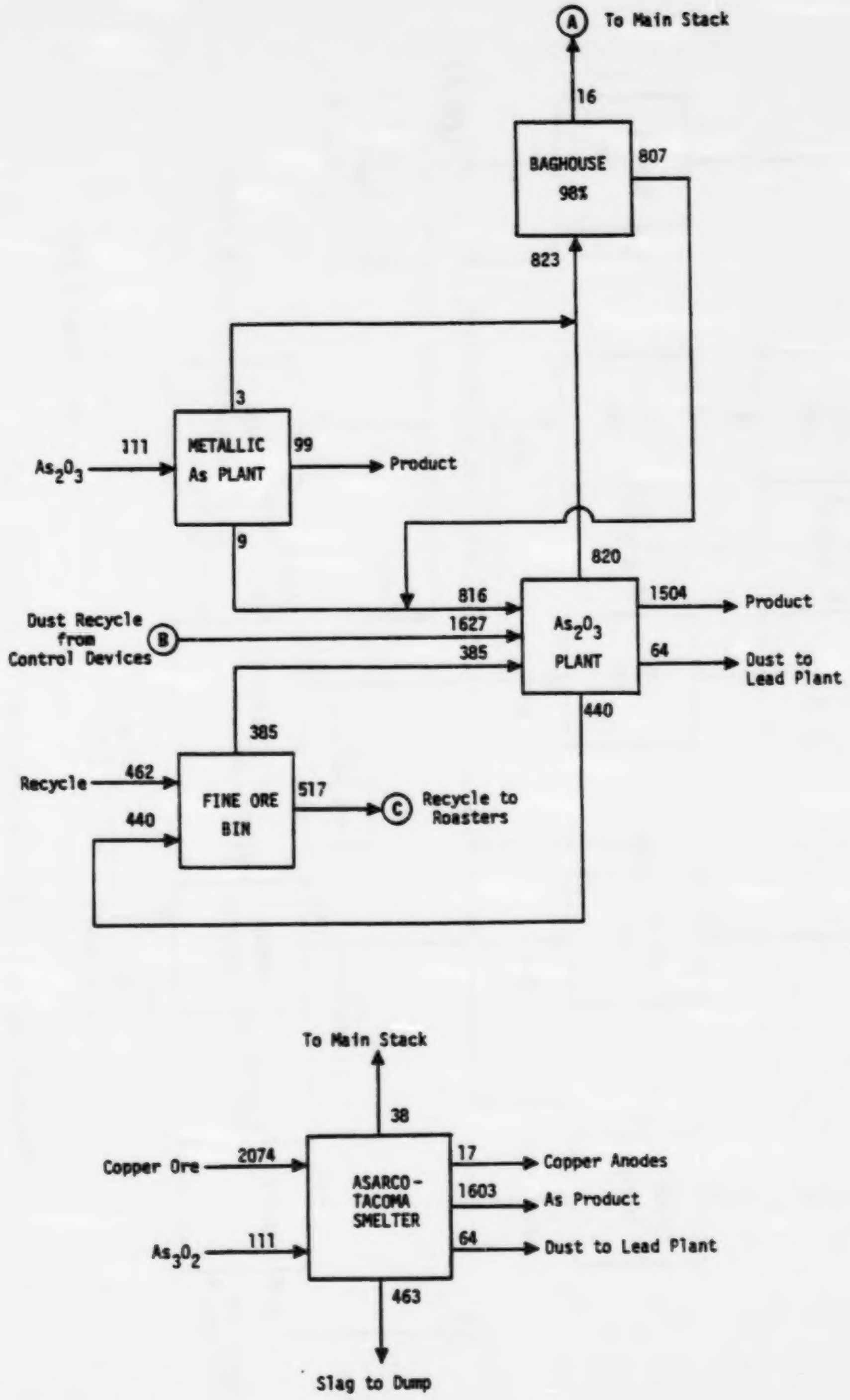

A
To Main Stack
16
BAGHOUSE
98%
807
823
3
111
METALLIC
As PLANT
99
Product
$As_2O_3$
9
820
816
1504
Product
Dust Recycle
from
Control Devices
B
1627
$As_2O_3$
PLANT
385
64
Dust to
Lead Plant
440
385
Recycle
462
FINE ORE
BIN
517
C
Recycle to
Roasters
440
To Main Stack
38
Copper Ore
2074
ASARCO-
TACOMA
SMELTER
17
Copper Anodes
1603
As Product
$As_3O_2$
111
64
Dust to Lead Plant
463
Slag to Dump

Figure 2-6. ASARCO-Tacoma Arsenic Material Balance (Concluded)

Table 2-4. SUMMARY OF PROCESS ARSENIC EMISSION ESTIMATES FOR ASARCO TACOMA

| Emission Source | Arsenic Content of Exit Gas | | Control Equipment | | Emission Rate | |
|---|---|---|---|---|---|---|
| | kg/hr | (lb/hr) | Type | Efficiency (percent) | kg/hr | (lb/hr) |
| Multihearth Roasters | 255 | (562) | Baghouse | 99.8 | 0.4 | (1) |
| Reverberatory Furnaces | 608 | (1341) | ESP | 98 | 9.5[a] | (21) |
| Converters[b] | 94 | (207) | ESP<br>ESP/acid plant<br>ESP/$SO_2$ plant | 96<br>99.9<br>99.9 | 0.04[c] | (0.1) |
| Anode Furnaces | 0.4 | (1) | ESP | 96 | 0.02 | (0.04) |
| Arsenic Plant | 373 | (823) | Baghouse | 98 | 7.3 | (16) |

[a] Of the potential arsenic emissions which leave the process, a portion is collected from the flue gas handling system as dust fallout prior to entering the control device indicated.

[b] Process offgases from the converters are separated into three separate streams, each directed to the control device(s) shown.

[c] The arsenic emission rate shown represents the sum of controlled emissions from the three control systems through which the converter offgases pass.

roasters, and the converters. Process emissions from the anode furnaces are very small compared to the other process emission sources. Controls are currently in place at ASARCO-Tacoma for all of the process emission sources. On a controlled basis, approximately 56 percent of the arsenic emissions from the ASARCO-Tacoma smelter in 1982 were emitted by the reverberatory furnaces, 42 percent were emitted by the arsenic plant, 3 percent were emitted by the multi-hearth roasters, and less than 1 percent were emitted by the converters and anode furnaces.

### 2.2.2 Fugitive Arsenic Emissions at ASARCO-Tacoma

Fugitive emissions may be characterized as emissions which escape directly to the atmosphere rather than through a flue or exhaust system. They result from leakage in and around process equipment and from material handling and transfer operations. These emissions may be considered as low level emissions as compared to process emissions, since they usually leave the smelter at or near ground level, whereas the process emissions are typically discharged through a tall stack.

Listed in Table 2-5 and shown in Figure 2-7 are potential sources of fugitive arsenic emissions. These emissions depend upon the particular types of equipment and operating practices employed by the copper smelter. To evaluate fugitive arsenic emission rates, EPA conducted tests at smelters where potential fugitive emission sources were controlled by high efficiency capture systems.[20,22] Measurements were made of the quantity of arsenic contained in the captured airstream. By assuming that the quantity of captured arsenic represented the fugitive emissions from an uncontrolled source, fugitive arsenic emission factors were developed for roasters, smelting furnaces, and converters. Fugitive arsenic emission factors for anode furnaces and miscellaneous sources were developed based on observation of the sources and on engineering judgment.

#### 2.2.2.1 Roaster.

Charging. Fugitive emissions during charging of multi-hearth roasters seldom occur because of the water content (8 to 10 percent) in the feed, and the "choke feed" mechanism used on the charging hoppers.

Leakage. Fugitive arsenic emissions from multi-hearth roasters may be emitted from leaks that can occur at the doors located at each

Table 2-5. POTENTIAL SOURCES OF FUGITIVE ARSENIC EMISSIONS

- Roaster
  - Charging
  - Leakage
  - Hot calcine discharge and transfer
- Smelting Furnace
  - Charging
  - Leakage
  - Matte tapping
  - Slag tapping
  - Converter slag return
- Converters
  - Charging (matte, reverts, flux, lead smelter by-products, cold dope)
  - Blowing (primary hood leaks)
  - Skimming
  - Holding
  - Pouring of slag and blister
  - Converter leaks
- Anode Furnace
  - Charging
  - Blowing
  - Holding
  - Pouring
- Miscellaneous
  - Dust handling and transfer
  - Ladles (matte and slag)
  - Slag dumping
  - Arsenic building (ASARCO-Tacoma smelter only)

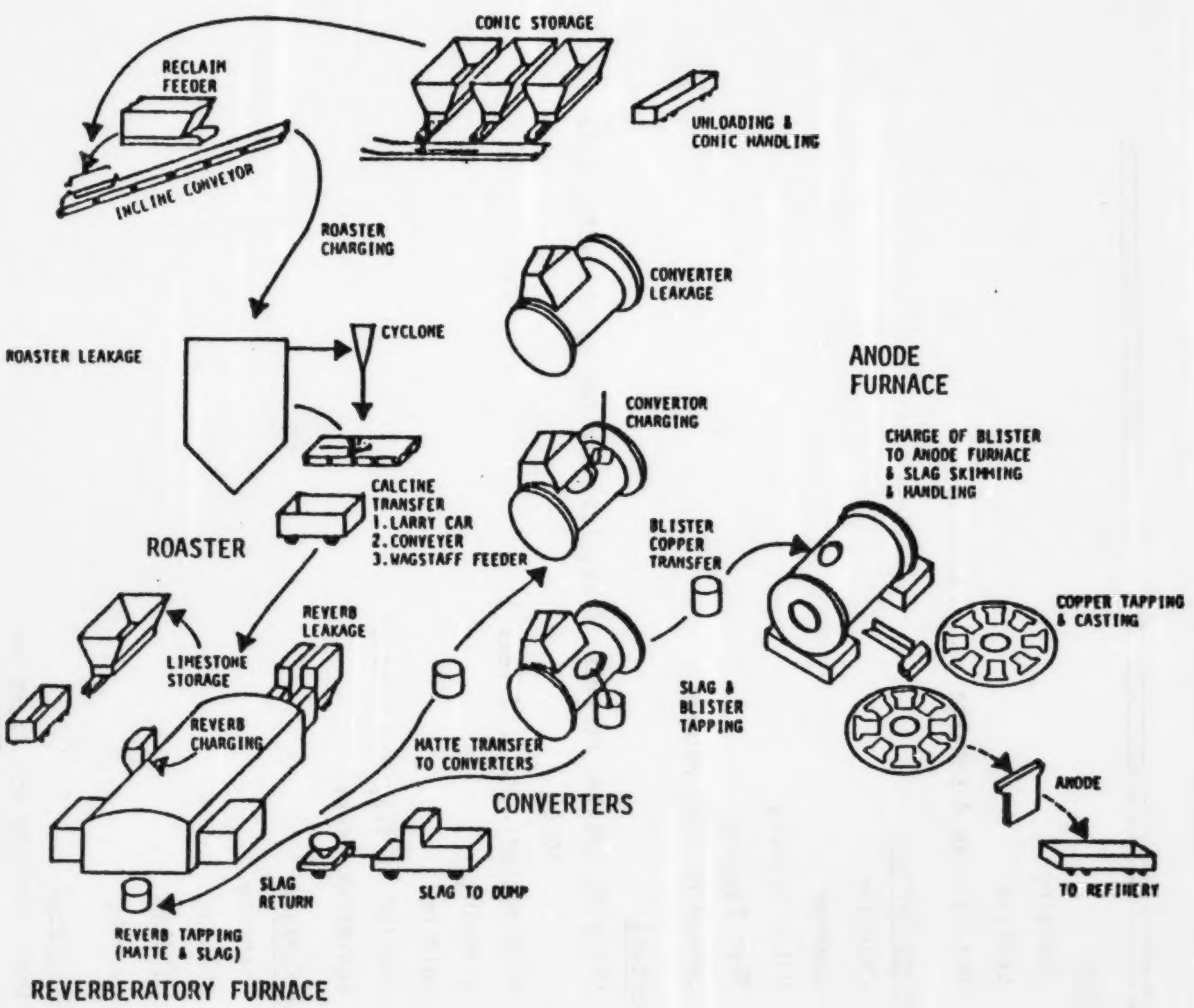

CONIC STORAGE
RECLAIM FEEDER
UNLOADING & CONIC HANDLING
INCLINE CONVEYOR
ROASTER CHARGING
CONVERTER LEAKAGE
ROASTER LEAKAGE
CYCLONE
ANODE FURNACE
CONVERTOR CHARGING
CHARGE OF BLISTER TO ANODE FURNACE & SLAG SKIMMING & HANDLING
CALCINE TRANSFER 1.LARRY CAR 2.CONVEYER 3.WAGSTAFF FEEDER
ROASTER
BLISTER COPPER TRANSFER
COPPER TAPPING & CASTING
REVERB LEAKAGE
LIMESTONE STORAGE
LIMESTONE UNLOADING
REVERB CHARGING
SLAG & BLISTER TAPPING
MATTE TRANSFER TO CONVERTERS
ANODE
CONVERTERS
SLAG RETURN
SLAG TO DUMP
TO REFINERY
REVERB TAPPING (MATTE & SLAG)
REVERBERATORY FURNACE

Figure 2-7. Fugitive Emission Sources at Primary Copper Smelters

one of the hearth levels, from holes in the actual shell of the roaster, or from leaks around the shaft that holds the rabble arms. Under normal operating conditions, these emissions are minimized by operating the multi-hearth roasters under a slight negative pressure required to supply induced air for oxidation.

Hot calcine discharge and transfer. Fugitive emissions may be generated during discharging and transfer of hot calcine from roaster to smelting furnaces. Smelters with multi-hearth roasters usually use larry cars (small rail cars) to transport calcines to the furnace. When the material is dropped from the calcine hopper located under the roaster into the covered car through a feed opening, large quantities of dust are generated as a result of material movement and pressure changes within the car. Some fugitive emissions can also occur during the transportation of the roaster calcines to the smelting furnace. In the case where larry cars are used, their feed opening is usually covered to minimize this effect.

Table 2-6 presents fugitive arsenic emission test data for the calcine discharge operation. It was assumed that complete capture of calcine discharge fugitive emissions was achieved by the capture system. Based on the test results, the average fugitive arsenic emission factor was determined to be 0.03 percent of the arsenic in the calcine.

### 2.2.2.2 Smelting furnace.

Leakage. The reverberatory furnace at ASARCO-Tacoma, like any other reverberatory furnace, is operated under a slight negative pressure. Since gases flow from the higher pressure environment to the lower pressure environment, the pressure differential between the gases inside the furnace and the air outside the furnace prevents the gases from escaping to the atmosphere through openings and cracks in the furnace walls. Pressure controls are used to minimize pressure surges as well as to maintain negative pressures. It is therefore reasonable to assume that fugitive arsenic emissions due to leakage from the smelting furnace are negligible.

Matte tapping. Matte tapping is a principal fugitive emission source at the smelting furnace. Smelting furnaces have from one to

## Table 2-6. CAPTURED FUGITIVE ARSENIC EMISSIONS FROM CALCINE DISCHARGE FROM MULTI-HEARTH ROASTERS AT ASARCO-TACOMA

| Sample run | Arsenic emissions,[a] kg/hr (lb/hr) | Sample time, min. | Calcine discharge during test time,[b] Mg (tons) | Arsenic content of calcine, % | Emission factor, % emitted from arsenic in calcine[c] |
|---|---|---|---|---|---|
| 1 | 0.46 (1.0) | 15 | 41.3 (45.5) | 0.7 | 0.04 |
| 2 | 0.76 (1.7) | 13 | 41.3 (45.5) | 1.22 | 0.03 |
| 3 | 1.24 (2.7) | 7 | 35.4 (39.0) | 1.22 | 0.03 |
| Avg. | 0.82 (1.8) | 11.6 | 39.3 (43.3) | 1.05 | 0.03 |

[a] Tests were performed only during calcine discharge.

[b] During first two tests, seven 6-1/2-ton larry cars were charged, and during the third test, six 6-1/2-ton larry cars were charged.

[c] Example calculation (Sample 1):

[0.46 kg/hr x (15 min ÷ 60 min/hr)] ÷ [(41.3 Mg calcine x 1,000 kg/Mg x 0.7 percent As)] x 100 = 0.04 percent As emitted.

three matte tap holes with associated launders on each side. Normally, only one tap is used at a time. The launder directs the flowing matte to a point where it can be collected in a large ladle. Fugitive emissions are observable from the point at which the matte leaves the furnace to the location where it enters the ladle. Typically, a matte tapping operation takes 5 to 15 minutes.

During matte tapping, the copper matte flows through a matte launder into a matte ladle. The matte tap holes, launders, and the matte ladle are ventilated by exhaust hoods. An arsenic emission estimate for matte tapping at ASARCO-Tacoma was developed based on the results of the arsenic emission measurements conducted by EPA and presented in Table 2-7.[20] It was assumed that complete capture of matte tapping fugitive emissions was obtained by the exhaust hood systems. The testing was performed only when actual matte tapping was being conducted. Based on the results, the average fugitive arsenic matte tapping emission factor was determined to be 2.5 percent of the arsenic in the matte.

Slag tapping. Slag tapping is another principal fugitive emission source at the smelting furnace. Slag tap ports and slag launders have been observed to emit less visible fugitive emissions (visible emissions are observed for a smaller percentage of the time it requires to perform a slag tapping operation) than those emitted during matte tapping operations.

At the ASARCO-Tacoma smelter, the slag tap hole, slag launder, and the slag pot are provided with exhaust hoods. Results of tests conducted by EPA to determine fugitive arsenic emissions during slag tapping are presented in Table 2-8.[20] It was assumed that all of the fugitive arsenic emissions were captured by the exhaust hoods. Based on the test results, the average fugitive emission factor for slag tapping was determined to be 0.12 percent of the arsenic in the slag.

Converter slag return. Converter slag is returned to the reverberatory furnace through the converter slag return launder. This is a simple channel with an opening in the furnace wall.

## Table 2-7. CAPTURED FUGITIVE ARSENIC EMISSIONS DURING MATTE TAPPING FROM THE REVERBERATORY FURNACE AT ASARCO-TACOMA

| Sample run | Arsenic emissions,[a] kg/hr(lb/hr) | Sample time, min. | Matte charged during test time, Mg (tons) | Arsenic content of matte, % | Emission factor, % emitted from arsenic in matte[b] |
|---|---|---|---|---|---|
| 1 | 1.69 (3.73) | 78 | 113.9 (125.6) | 0.23 | 0.84 |
| 2 | 7.53 (16.60) | 75 | 113.9 (125.6) | 0.25 | 3.30 |
| 3 | 7.37 (16.26) | 74 | 113.9 (125.6) | 0.23 | 3.47 |
| Avg. | 5.53 (12.20) | 76 | 113.9 (125.6) | 0.24 | 2.54 |

[a] Testing was performed only during actual matte tapping.

[b] Example calculation (Sample 1):

[1.69 kg/hr x (78 min ÷ 60 min/hr)] ÷ [(113.9 Mg matte x 1,000 kg/Mg x 0.23 percent As)] x 100 = 0.84 percent As emitted.

### Table 2-8. CAPTURED FUGITIVE ARSENIC EMISSIONS DURING SLAG TAPPING FROM THE REVERBERATORY FURNACE AT ASARCO-TACOMA

| Sample run | Arsenic emissions,[a] kg/hr(lb/hr) | Sample time, min. | Slag tapped during test time,[b] Mg (tons) | Arsenic content of slag, % | Emission factor, % emitted from arsenic in slag[c] |
|---|---|---|---|---|---|
| 1 | 0.25 (0.55) | 120 | 229 (252) | 0.33 | 0.066 |
| 2 | 0.65 (1.43) | 111 | 211 (233) | 0.38 | 0.162 |
| 3 | 0.53 (1.18) | 60 | 114 (126) | 0.38 | 0.122 |
| Avg. | 0.48 (1.05) | 97 | 185 (204) | 0.36 | 0.117 |

[a] Tests were performed only during actual slag tapping.

[b] Slag is tapped in 6.3-ton ladles. It takes about 15 minutes to fill a 5-pot slag train.

[c] Example calculations (Sample 1):

[0.25 kg/hr x (120 min ÷ 60 min/hr)] ÷ [(229 Mg slag x 1,000 kg/Mg x 0.33 percent As)] x 100 = 0.066 percent As emitted.

The number of times converter slag is returned to the furnace depends upon the number of converters and the operating level of the smelter. Each time the slag is returned, pressure fluctuations occur in the furnace. These fluctuations are due to the agitation in the bath and the rapid chemical reactions among the slag constituents. This tends to generate fugitive emissions through the relatively large converter slag opening.

Emission test results for converter slag return operations at ASARCO-Tacoma are presented in Table 2-9.[20] Since converter slag return operations are of short duration, 1 to 3 minutes, testing was performed for 3 days to obtain an adequate sample for analysis. The results indicate that about 0.03 percent of the arsenic in the converter slag is released as fugitive emissions.

2.2.2.3 Converter.

Charging. During converter charging, the converter is tilted by a drive mechanism until the mouth of the converter is approximately 45 degrees from the vertical. Fugitive emissions during this operation result when matte or other materials such as reverts flux are poured from a ladle into the converter. The gate on the primary hood is retracted to its highest position. An overhead crane lifts the matte ladle above the mouth of the converter and pours the matte into the converter by tilting the ladle. During this operation, visible emissions are heavy, but of short duration. When charging is completed, the blowing air is turned on while the tuyeres are above the surface of the molten bath. The converter is rotated until its mouth is in the topmost position, contained within the primary hood. The gate is then lowered and the blowing cycle commences.

Blowing. Most domestic smelters have attempted to provide relatively close fitting primary hoods on converter openings. These hoods are used to contain and capture offgases generated during blowing operations. However, these hoods at best do not completely seal the opening since sufficient space has to be provided for easy movement of the converter shell. This is to allow for converter shell irregularities and the molten copper buildup on the shell due to splashing during the blowing operation. Some fugitive emissions are discharged through

## Table 2-9. CAPTURED FUGITIVE ARSENIC EMISSIONS DURING CONVERTER SLAG RETURN AT ASARCO-TACOMA

| | |
|---|---|
| Arsenic emission rate, kg/hr (lb/hr) | 0.13 (0.3) |
| Sample time, minutes | 22.92 |
| Converter slag returned during sample time, Mg (tons)[a] | 103 (113) |
| Arsenic content of slag return, % | 0.187 |
| Arsenic emission factor, % emitted from the arsenic content of slag return[b] | 0.03 |

---

[a] Eighteen 6.3-ton converter slag ladles were returned during the test time.

[b] Calculation:

[0.13 kg/hr x (22.92 min ÷ 60 min/hr)] ÷ [(103 Mg converter slag return x 1,000 kg/Mg x 0.187 percent As)] x 100 = 0.03 percent As emitted.

these openings, especially when pressure surges occur during converter blowing, and rapid adjustments are not made by the duct damper system.

*Skimming*. During skimming operations, the mouth of the converter is rotated to a position between 65 and 85 degrees from the vertical, depending upon the bath level. The blowing air remains on as the converter is rolled out, until the tuyeres are above the surface of the bath. This action results in significant quantities of fugitive emissions, these openings, especially when pressure surges occur during converter but is necessitated by the equipment configuration. Slag is skimmed from the converter mouth into a slag ladle. Fugitive emissions are visible during this operation. The primary hood gate may not be fully retracted during skimming; however, the hood is isolated by dampers from the main duct system to keep dilution air from mixing with the high $SO_2$ offgases from blowing converters. At the completion of skimming, the converter mouth is again rotated to its vertical position within the hood. The gate is fully extended, and the blowing cycle resumes.

*Pouring*. As the blister copper is poured, the converter is slowly rotated downward until its mouth reaches a position approximately 90 to 125 degrees from the vertical, depending upon the size of the pour and the buildup within the converter. The hood gate may be partially extended during this operation. Fugitive emissions occur as the blister copper is poured from the converter into the ladle. After the blister pour, the converter mouth is rotated upward to a position approximately 45 degrees from the vertical to await a new matte charge and the start of a new cycle.

*Holding*. There are times during normal smelting operations when material, either slag or blister copper, cannot be immediately transferred from the converters to the ladles. This may be due to conditions such as unavailability of the crane, refining furnace operations not allowing additional feed material, or others. It then becomes necessary for the converter to be placed in a holding mode. In this mode the converter is rotated until the mouth of the converter is 30 to 45 degrees from the vertical to keep the tuyeres out of the bath. This results in fugitive emissions from the molten material in the converter being discharged into the converter building.

*Converter leaks*. Since the ends of most Pierce-Smith converters are joined by bolts and springs, they occasionally leak at the end joint. When this leakage is located below the molten material surface, it is usually repaired rapidly to prevent major erosion. However, in those cases where it is located above the charge surface, it may not be repaired. Thus, fugitive emissions may occur at this point.

Fugitive Emission Rate. Tests were performed to estimate fugitive arsenic emissions from a single converter at the ASARCO-Tacoma smelter.[22] The tests were conducted by sampling the fugitive emission plume directly above the converter. Mass emission rates of arsenic in the fugitive plume were obtained by applying a diffusion model. Results obtained in two test runs were 0.39 kg/hr (0.87 lb/hr) and 0.79 kg/hr (1.73 lb/hr) of fugitive arsenic mass rate. These tests were performed under nonisokinetic conditions (sampling rates during the two runs were 351 and 155 percent of isokinetic, respectively). In addition, simultaneous tests were performed in the primary converter offgas duct for process arsenic emissions in order to determine the mass balance for arsenic over a converter cycle. No balance could be obtained because substantial quantities of arsenic from the converter, about 150 kg/hr (330 lb/hr), were unaccounted for. These substantial quantities of unaccounted arsenic indicate that larger amounts of fugitives than those indicated by the fugitive emission test data may be emitted to the atmosphere. Data from the entire test were determined to be inconclusive and, therefore, were not used to estimate the fugitive emission rate.

EPA also performed testing at the ASARCO-El Paso smelter[21] to estimate fugitive arsenic emissions occurring during converting. Inlet and outlet measurements were made across the baghouse which treats the gases from the converter building evacuation system (see Section 3.3.2.3). The results indicated a fugitive arsenic emission factor for converters of 15 percent of the arsenic contained in the converter primary offgas. This factor was used for estimating converter fugitive emissions at ASARCO-Tacoma.

2.2.2.4 Anode furnace. Refining of blister copper to anode copper at ASARCO-Tacoma is performed in hearth-type furnaces. The process offgases generated during the actual refining (oxidation and reduction blows) are siphoned through a furnace offtake and ducted to a cold electrostatic precipitator. Some fugitive emissions are emitted during charging, skimming, poling, and casting operations.

No test data are available for determining the amount of fugitive arsenic emissions from anode furnaces. Therefore, an approach analogous to the one used to estimate converter fugitive emissions was selected. The anode furnace fugitive emission estimate for the ASARCO-Tacoma smelter is based on the amount of arsenic present in the anode furnace process offgases as presented in the arsenic material balance (Figure 2-6). Using engineering judgment, an emission factor of 20 percent of the arsenic contained in the anode furnace process offgas was assumed for estimating the anode furnace fugitive emissions at ASARCO-Tacoma.

#### 2.2.2.5 Miscellaneous and fugitive emission sources.

Dust handling and transfer. Dust handling and transfer can generate fugitive emissions if carelessly performed. However, most smelters take reasonable precautions to minimize fugitive emissions from dust handling and transfer. Dust transfer from control devices onto conveyors is typically performed by mechanically timed and activated rotary valves or twin gravity self-closing gates. These conveyors are generally covered by housings and discharge into storage bins from which dust may be withdrawn as desired. Dust transfer from storage bins is usually through dust-tight connections to surface transportation units such as tank trucks and dumpsters. Cleaning and unloading of dust from flues and settling chambers is performed by conveyors which feed into hoppers provided at spaced intervals underneath the flues and settling chambers. Both screw and drag-type conveyors are used. These flue dusts are usually treated in a pugmill or pelletizing disc where moisture is added. The wet dust is then transferred to a bedding area, blended with other feed constituents, and recycled. Dust from waste heat boilers and crossover flues is usually removed by hand methods, water-bomb lances, slugger guns, and other methods. This dust is handled and transported by surface vehicles to the smelter flux crushing system. For the purpose of developing an arsenic emission factor for these operations, it was assumed that all transfer systems such as screw or drag type conveyors are dust-tight. The uncontrolled fugitive arsenic emissions from these systems were assumed to not exceed 0.1 percent of the arsenic in the dust collected at arsenic drop-out points such as control device storage hoppers, smelter flues, dust chambers, and waste heat boilers. This emission factor was based on particulate (dust) emission rates found in grain transfer, asphalt batching, and other industries involved in transfer, handling, and conveying operations. In these industries, 0.1 percent of the dust transferred, handled, or conveyed is estimated to appear as fugitives.[23]

Ladles. Normal process fluctuations may require that ladles containing matte, converter slag, or blister copper be temporarily set aside until needed. Because these ladles contain molten material, some emissions can be observed due to fuming. These, however, are short-lived. The exposed surface of the material cools rapidly, forming a solidified layer or skull which greatly limits fugitive emissions. It was therefore assumed that no fugitive emissions are discharged from ladles once they are filled.

Slag dumping. Smelting furnace slag is disposed of by water granulation or by transport in the molten state for dumping a short distance from the smelter. Slag dumping is the more widely used method. The slag is transported to the dump site by train or slag hauler. The slag train is usually comprised of a number of slag pots or ladles on flat cars. Solidification at the surface of the slag in the pots is fairly rapid. Fugitive emissions during transportation to the dumping site are therefore limited. However, during dumping of slag at the dumping site, substantial fugitive emissions, although short in duration (less than 1 minute), can be observed.

Testing was performed by EPA at the ASARCO-Tacoma smelter to determine the magnitude of fugitive arsenic emissions during slag dumping.[20] Reverberatory slag from slag pots was analyzed for arsenic content on exit from the reverberatory furnace and after the slag was deposited on the dump site. Process sample analysis was performed on four separate days. The test results obtained are listed in Table 2-10.

### Table 2-10. REVERBERATORY FURNACE SLAG ANALYSIS FOR ARSENIC CONTENT AT ASARCO-TACOMA

| Sample Run | Percent arsenic in slag at exit from furnace slag launder | Percent arsenic in slag at dump site |
|---|---|---|
| 1 | 0.33 | 0.40 |
| 2 | 0.38 | 0.52 |
| 3 | 0.46 | 0.44 |
| 4 | 0.49 | 0.29 |
| Avg. | 0.42 | 0.41 |

Test results indicated that in the first two of the four samples the arsenic content of the reverberatory furnace slag increased in concentration at the dump site. In the third sample there was a slight decrease in concentration at the dump site. The fourth sample indicated a larger decrease in concentration at the dump site. However, comparison of the averaged results indicated approximately the same concentration of arsenic in the slag at the exit from the furnace and at the dump site. This indicates that fugitive arsenic emissions during slag dumping can be considered negligible.

Arsenic Building. The ASARCO-Tacoma smelter is the only domestic copper smelter that operates an arsenic plant and currently manufactures arsenic for commercial use. Dust collected in the flues, baghouse, and electrostatic precipitators serving the roasters and the reverberatory furnaces is conveyed to one of the six Godfrey roasters in the arsenic building. In these roasters, the arsenic-containing dust is heated to vaporize the arsenic trioxide. The vapors then pass through a baffled brick chamber, or "kitchen," where they cool, condense, and drop out in a granular form. The chambers are opened periodically, and the condensed arsenic trioxide is removed manually. Fugitive arsenic escapes during this cleaning period. Fugitive arsenic emission estimates for the arsenic building at the ASARCO-Tacoma smelter were based on information obtained from the Puget Sound Air Pollution Control Agency

(PSAPCA).[24] PSAPCA estimates indicate that the major source of fugitive arsenic emissions in the arsenic building are the arsenic kitchens. They estimated that 0.5 percent of the arsenic when pulled from the arsenic kitchens becomes airborne. This emission factor was therefore used to estimate the fugitive arsenic emissions from the arsenic building.

2.2.2.6 Summary of fugitive arsenic emission estimates. Table 2-11 presents a summary of fugitive arsenic emissions estimated for ASARCO-Tacoma. These estimates are based on emission factors described in the previous sections and arsenic inputs to various process steps presented in Figure 2-6. It is estimated that 90 percent of the total fugitive arsenic emissions at the ASARCO-Tacoma smelter is emitted by the converters.

## Table 2-11. SUMMARY OF FUGITIVE ARSENIC EMISSION ESTIMATES FOR ASARCO-TACOMA

| Emission source | Arsenic Content | | Emission factor, % emitted from input arsenic | Emission rate in absence of control kg/hr (lb/hr) | Control Equipment | Control Efficiency (percent) | | | Emission rate, kg/hr (lb/hr) |
|---|---|---|---|---|---|---|---|---|---|
| | Present in | Amount, kg/hr (lb/hr) | | | | capture | collection | overall | |
| Calcine transfer from multi-hearth roasters | Calcine | 920 (2,029) | 0.03 | 0.3 (0.6) | baghouse | 90 | 99.8 | 89.8 | 0.03 (0.07) |
| Matte tap | Matte | 148 (327) | 2.5 | 3.8 (8.4) | ESP | 90 | 96 | 86.4 | 0.5 (1.1) |
| Slag tap | Furnance slag | 210 (463) | 0.12 | 0.25 (0.6) | ESP | 90 | 96 | 86.4 | 0.03 (0.08) |
| Converter slag return | Converter slag | 46.3 (102) | 0.03 | 0.01 (0.02) | none | - | - | - | 0.01 (0.02) |
| Converter operation | Process offgas | 94 (207) | 15 | 14 (30.9) | none | - | - | - | 14 (30.9) |
| Anode furnaces | Blister copper | 0.4 (1) | 20 | 0.08 (0.2) | none | - | - | - | 0.08 (0.2) |
| Miscellaneous (Flue dust transfer, handling, and conveying) | Dust transferred, handled, and conveyed | 2,688 (5927) | 0.1 | 2.7 (5.9) | a | a | a | 90 | 0.3 (0.6) |
| Arsenic building | Charge to arsenic plant | 1,282 (2828) | 0.5 | 6.4 (14.1) | b | b | b | 90 | 0.6 (1.4) |

[a] Flue dust transfer, handling, and conveying emissions are estimated to be 90 percent controlled by means of dust-tight transfer systems.

[b] Arsenic building emissions are estimated to be 90 percent controlled by means of dust control work practices.

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# 3.0 CONTROL TECHNOLOGY

This chapter identifies alternative control techniques which can be applied to process and fugitive emission sources at primary copper smelters to control arsenic emissions. The amount of arsenic emission reduction which can be achieved by the application of these control techniques as well as major factors affecting their performance are discussed.

## 3.1 ALTERNATIVE CONTROL TECHNIQUES

### 3.1.1 Process Emission Controls

3.1.1.1 General Considerations. As discussed in Section 2.0, much of the arsenic entering the copper smelting process is volatilized and eliminated as metallic oxide in the process offgas streams. This is a result of the very high temperatures associated with the pyrometallurgical processes used and the inherent volatility of arsenic and its prevalent oxide, arsenic trioxide ($As_2O_3$). The extreme volatility of arsenic, and especially arsenic trioxide, is the single most important factor affecting the controllability of arsenic emissions from sources at primary copper smelters.

Several studies have been made to determine the vapor pressure of arsenic trioxide in air at various temperatures.[1] Table 3-1 presents vapor pressure data for arsenolite ($As_4O_6$), the more common form of arsenic trioxide and the most abundant arsenic compound in smelter offgases.[2,3] Also presented are the arsenic concentrations at saturation corresponding to the temperatures and vapor pressures listed. Figure 3-1 illustrates graphically the vapor pressure-temperature relationship indicated by these data. This figure illustrates a significant logarithmic increase in the vapor pressure of arsenic trioxide, and thus the amount of arsenic which can exist in the vapor state, with temperature.

Table 3-1. SUMMARY OF $As_4O_6$ VAPOR PRESSURE DATA AND CORRESPONDING ARSENIC CONCENTRATION AT VARIOUS TEMPERATURES

| Temperature, °C(°F) | Vapor pressure, mm Hg | Arsenic concentration, $g/m^3$ |
|---|---|---|
| 457.2 (855.0) | 760 | $5.0 \times 10^3$ |
| 412.2 (774.0) | 400 | $2.81 \times 10^3$ |
| 332.5 (630.5) | 100 | $7.94 \times 10^2$ |
| 299.2 (570.6) | 40 | $3.36 \times 10^2$ |
| 259.7 (499.5) | 10 | 90.20 |
| 212.5 (414.5) | 1.0 | 9.90 |
| 200.0 (392.0) | 0.910 | 9.25 |
| 175.0 (347.0) | 0.175 | 1.88 |
| 150.0 (302.0) | $2.77 \times 10^{-2}$ | 0.315 |
| 125.0 (257.0) | $3.58 \times 10^{-3}$ | 0.043 |
| 110.0 (230.0) | $8.81 \times 10^{-4}$ | 0.011 |
| 100.0 (212.0) | $3.31 \times 10^{-4}$ | 0.0043 |
| 90.0 (194.0) | $1.18 \times 10^{-4}$ | 0.0016 |

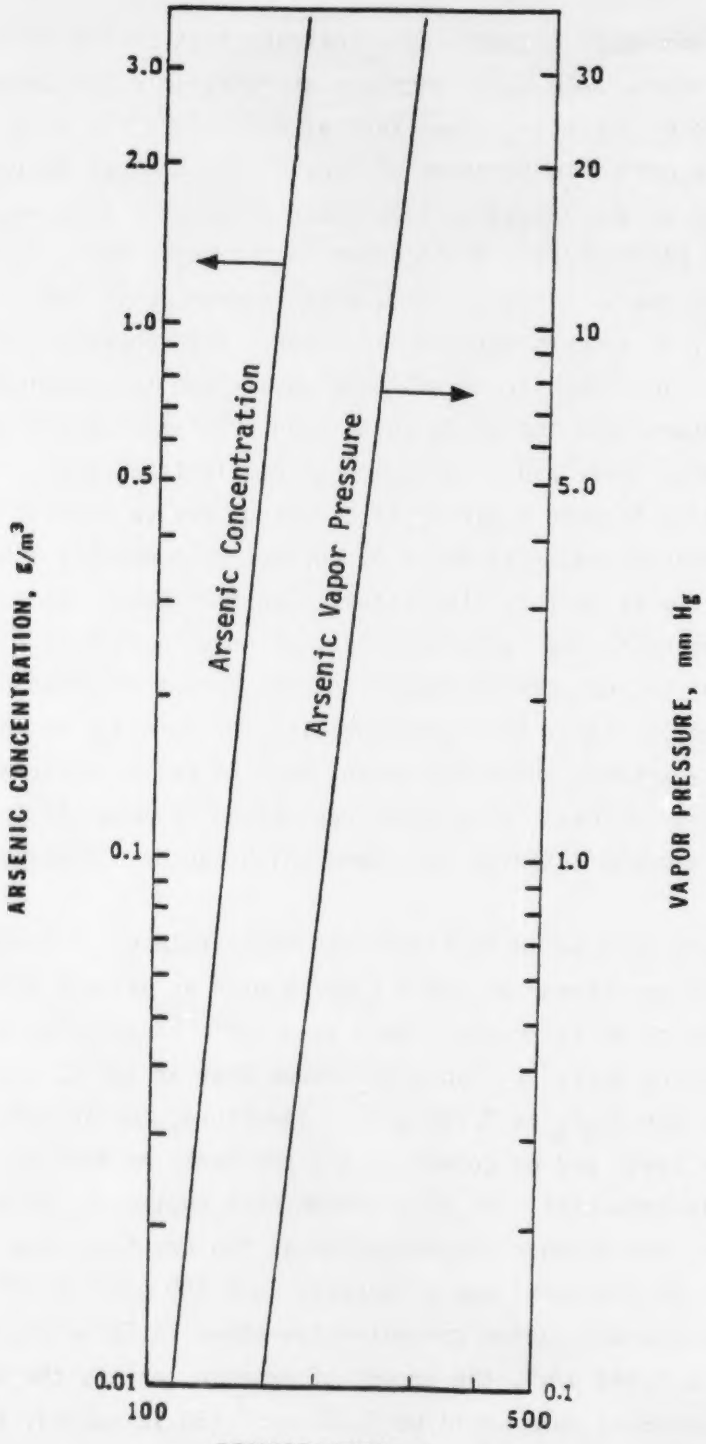


Figure 3-1. Arsenic Trioxide Vapor Pressure and Saturated Vapor Concentration with Temperature

Furthermore, the vapor pressure data indicate that arsenic trioxide maintains an appreciable vapor pressure at relatively low temperatures.

For example, Table 3-1 shows that at 200°C (392°F), which is typical of the outlet temperature of many of the control devices currently used in the industry, 9.25 grams of arsenic trioxide per $m^3$ of gas ($g/m^3$) (4.04 $gr/ft^3$) could exist in the vapor form. Even at a temperature as low as 125°C (257°F), a gas stream could contain 0.043 $g/m^3$ (0.019 $gr/ft^3$) of arsenic trioxide as vapor. Any concentration higher than 9.25 $g/m^3$ in a gas stream at 200°C would lead to condensation of some of the vapor, and the $As_4O_6$ would then exist in two phases: a vapor or gaseous phase and a condensed or particulate phase. The vapor would pass through a particulate control device without collection, while the condensed material would be collected at about the same efficiency as total particulate matter. The implication of these facts with regard to the controllability of arsenic is important. The temperature of the gas stream determines the amount of arsenic trioxide which can exist as vapor and, consequently, the quantity of condensed or particulate arsenic which can potentially be collected in a particulate control device. A lower temperature gas stream is more likely to contain more arsenic trioxide in a form which can be collected in such a device.

An example will serve to illustrate these points. A 2,834 $m^3$/min (100,000 acfm) gas stream at 150°C (302°F) with an arsenic trioxide mass flow rate of 40 kg/hr would have an arsenic concentration of 0.235 $g/m^3$ (0.103 $gr/ft^3$). Table 3-1 shows that at 150°C, the saturation concentration for $As_4O_6$ is 0.315 $g/m^3$. Therefore, the stream is at a subsaturation level and no condensation, and hence no particulate collection, is predicted. If this stream were cooled to 125°C in a spray chamber, the arsenic concentration at the new flow rate of 2,667 $m^3$/min (94,100 acfm) would increase to 0.250 $g/m^3$ (0.109 $gr/ft^3$). Since the arsenic saturation concentration shown in Table 3-1 for this temperature is 0.043 $g/m^3$, the amount of arsenic leaving the spray chamber in condensed form would be 0.207 $g/m^3$ (33.12 kg/hr), and the remaining amount, 0.043 $g/m^3$, would remain in vapor form. If this stream was then sent to a baghouse collector with a control efficiency for total particulate matter of 96 percent, a total of 0.96 x

33.12 kg/hr = 31.80 kg/hr would be collected. The overall control efficiency for arsenic at the lowered temperature would thus be 31.80/40 = 80 percent. From Table 3-1 it can be seen that as the temperature is further lowered below 125°C, the saturation concentration becomes very small. In this case, virtually all of the arsenic trioxide present in the stream would condense into a collectible form and the collection efficiency for arsenic would approach the particulate efficiency of 96 percent.

Tests conducted by EPA across a hot electrostatic precipitator (ESP) which controlled particulate emissions from a "green" charge reverberatory smelting furnace further demonstrate this phenomenon.[4] The electrostatic precipitator was operated at 315°C (600°F) or higher. Measurements for particulate matter conducted using in-stack filters, at the operating temperature of the ESP, demonstrated an overall collection efficiency for particulate matter of about 97 percent. In contrast, arsenic measurements conducted using a modified EPA Method 5 sampling train operated at 121°C (250°F) indicated an average arsenic collection efficiency for the ESP of less than 30 percent (see Table 3-2). Calculations based on the arsenic concentration measured at the hot ESP inlet and the saturation concentration data presented in Table 3-1 indicate that the application of a 97 percent effective ESP to the same reverberatory furnace process gas stream which had been cooled to 110°C (230°F) could result in an overall arsenic collection efficiency of 90 percent. These measurements and calculations suggest that, while the subject ESP was reasonably effective in removing material which existed as particulate matter at its operating temperature, any material such as arsenic trioxide which would be predicted to exist in the vapor state at the elevated temperature at which the ESP was operated passed through the ESP with little removal. This demonstrates the need to cool the gas stream to be treated to the extent practicable to condense as much of the arsenic trioxide vapor as possible prior to its entering a control device for collection.

As a result, the alternative control techniques for arsenic process sources considered herein include precooling as an integral part of the overall control system. These include the application of baghouses, high voltage electrostatic precipitators, or high energy

Table 3-2. ARSENIC DATA FOR HOT ESP

| Sample run | Inlet | | | Outlet | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Temp., °C (°F) | Emissions | | Temp., °C (°F) | Emissions | | |
| | | g/Nm$^3$(gr/scf) | kg/hr (lb/hr) | | g/Nm$^3$ (gr/scf) | kg/hr (lb/hr) | |
| 1 | 328 (622) | 0.268 (0.117) | 26.8 (59.1) | 313 (595) | 0.211 (0.092) | 24.3 (53.6) | 9.3 |
| 2 | 317 (602) | 0.325 (0.142) | 33.0 (72.7) | 321 (610) | 0.181 (0.079) | 20.2 (44.6) | 38.6 |
| 3 | 336 (639) | 0.334 (0.146) | 34.1 (75.2) | 304 (580) | 0.199 (0.087) | 23.2 (51.1) | 32.0 |
| Avg. | 327 (621) | 0.309 (0.135) | 31.3 (69.0) | 313 (595) | 0.197 (0.086) | 22.6 (49.8) | 27.8 |

venturi scrubbers, preceded by gas stream cooling. In addition, these include contact sulfuric acid plants, with their extensive gas precleaning and conditioning systems, applied to process sources which generate offgases containing high concentrations of $SO_2$ (greater than 3.5 percent).

Before proceeding with the discussions on control techniques, it should be noted that the 30 percent arsenic collection efficiency recorded for the hot ESP cannot be predicted using the vapor pressure data for arsenic trioxide. In fact, given the operating temperature of the ESP (315°C), no arsenic recovery would be predicted because the arsenic concentration in the gas stream was several orders of magnitude lower than that needed to achieve saturation. This suggests (1) that other arsenic compounds less volatile than arsenic trioxide may be present in the gas stream, and/or (2) that condensation, although the principal mechanism in situations where more arsenic trioxide is present than is needed to saturate the gas stream, is not the only mechanism by which arsenic trioxide can be collected.

3.1.1.2 Gas Cooling. Methods commonly applied in the nonferrous metals industries for cooling hot gas streams include radiative cooling, evaporative cooling, and cooling by dilution with ambient air. The dilution method consists of introducing sufficient quantities of ambient air into the hot gas stream so that the resultant gas mixture is at the desired temperature. The ambient air required may be introduced by infiltration or forced draft. Evaporative cooling uses water to cool hot gases in spray or quench chambers. Water is injected into the hot gas stream where the heat contained in the gases vaporizes the injected water, resulting in a temperature reduction. Radiative cooling relies on heat loss due to natural convection and radiation to effect cooling. These losses occur whenever a temperature gradient exists between gases inside a duct and the surrounding air. Cooling of gases by this method requires only that a sufficient heat transfer area be available to obtain the desired cooling. All three methods have advantages and disadvantages.

Although cooling with dilution air is the simplest alternative, its application for the gas volumes and temperatures under consideration may not be economical. Depending on the temperature of the gas stream to be treated, the amount of dilution air needed to effect cooling could result in a two- to four-fold increase in the total gas volume to be treated, with a corresponding increase in the size and cost of the draft fan and the control device required.

Due to the need for sufficient heat transfer area, radiative cooling requires considerable space. Depending on the temperature of the gas stream to be cooled, the heat transfer areas required could exceed 1.6 $m^2/m^3$ (500 $ft^2/1000\ ft^3$) of gas treated. Normally used in the lead industry on zinc fuming operations, radiative cooling devices consist of a series of 10 to 20 U-shaped tubes, each about 3 m (10 ft) in diameter and 20 m (66 ft) in height. In addition, fan horsepower requirements increase due to the increased resistance to gas flow resulting from the added ductwork. The major drawback to radiative cooling, however, is its limited flexibility for temperature control.

Cooling hot gases by evaporative cooling is relatively simple and requires little space. Water spray chambers are currently used at a number of copper smelters for cooling process gases from a variety of sources prior to entering electrostatic precipitators or baghouses for particulate removal. Typically, the spray chambers have a cross-sectional area of about 35 $m^2$ (375 $ft^2$) and are 30 to 60 m (100 to 200 ft) in length. The large cross-sectional area results in a low flow velocity and a relatively long residence time. Water is introduced through a series of sprays along the cross section of the chamber. Water requirements vary depending on the temperature of the stream to be cooled and the desired end temperature.

The major difficulty in applying water spray chambers for cooling smelter offgases is the potential for corrosion. The amount of cooling achievable is limited by the dew point of the treated gas stream. As the gases are cooled, water and sulfuric acid mist contained in the gas stream may condense, creating a corrosion problem in the flue system, control device, and stack. While the problem of corrosion attack is potentially severe, it can be negated by the use of appropriate construction materials. Acid resistant cement, stainless steel, nickel, and chromium alloys have been used in several cases. No significant corrosion problems have been reported at primary copper and lead smelters where spray chambers are used for cooling prior to electrostatic precipitators and baghouses used for particulate matter control.

3.1.1.3 Baghouses (Fabric Filters). Baghouse particulate collectors have historically achieved collection efficiencies in excess of 99 percent, over a broad range of applications. Although extensively used in the primary lead and zinc industries for the collection of particulate and metallurgical fume, their application at primary copper smelters has been limited. ASARCO-Tacoma is the only smelter which currently uses a baghouse for the control of particulate matter contained in smelter offgases.

In principle, particles contained in the treated gas stream are initially captured and retained on the fibers of the fabric by a number of mechanisms, including direct interception, inertial impaction, diffusion, gravitational settling, and electrostatic attraction. Once a mat or cake of dust is formed on the fabric, further collection is achieved by simple sieving. Periodically, the fabric is cleaned by mechanical or other means to allow for disposal of the collected particulate and to maintain the pressure drop across the filter within practical operating limits.

The filtration area required at a specified pressure drop is dependent on the gas volume treated, particulate loading, permeability of the fabric used, resistance properties of the particulate deposited, and the cleaning mechanism used. The pressure drop commonly found in baghouses applied in the nonferrous metals industries ranges from 0.5 to 2.0 kPa (2 to 8 inches $H_2O$).[5] The filtering velocity, or air-to-cloth ratio, generally ranges from 0.30 to 0.61 $m^3$/min per $m^2$ (1 to 2 acfm/$ft^2$) for conventional mechanical shaker cleaning type baghouses when applied to metallurgical fume. Pulse jet cleaning type units generally operate at higher air-to-cloth ratios, ranging from 1.8 to 3.0 $m^3$/min per $m^2$ (6 to 10 acfm/$ft^2$).[5]

Although there is a variety of woven and felted fabrics available, woven Orlon and Dacron bags should be suitable in smelter applications. Both exhibit good resistance to acid attack and may be operated at temperatures ranging up to 135°C (275°F) with no significant deterioration.[5]

Bag life, which varies considerably with operating conditions, should be approximately 1 to 3 years. Closed pressure or closed suction baghouse designs may be used in the applications considered. The baghouse design selected depends on the acid dew point and the consequent corrosion potential of the smelter offgases being treated.

3.1.1.4 Electrostatic Precipitators (ESP's). Single-stage electrostatic precipitators are widely used in the primary copper industry for the control of particulate emissions from smelting facilities. Electrostatic precipitators use electrical forces for the removal of suspended particulates in a gas stream. The process encompasses three basic functions: the charging of particles, the collection of charged particles, and the removal of the collected particles. Particles suspended in the gas stream are charged while passing through a high voltage, direct current corona established between a discharge electrode, usually a small diameter wire which is maintained at high voltage, and a grounded collecting surface (collecting electrode). As the particles pass through the corona, they are bombarded by negative ions emanating from the discharge electrode, and charged within a fraction of a second. The charged particles, influenced by electric field forces, migrate toward the grounded collecting surface where they are deposited and held by electrical, mechanical, and molecular forces. Particulate matter adhering to the collecting surface is periodically dislodged by mechanical rappers or by flushing with water. The material is collected in a hopper and periodically removed for disposal or recycle.

Two principal types of precipitator design are available, the wire-and-plate and wire-in-tube types. In the more common plate-type precipitators, the collecting surface consists of parallel vertical plates spaced 15 to 30 cm apart with wire or rod discharge electrodes suspended vertically between the plates. The plates are typically 4 to 12 m in height and 4 to 7 m in length.[6] Plate-type ESP's are generally applied directly to process sources for the control of dry particulate matter at elevated temperatures, usually 200 to 340°C (400 to 650°F). The collecting surface in tube-type precipitators consists of a cylinder with the discharge electrode centered along its longitudinal axis. This type of ESP is used exclusively for acid mist elimination and fine particulate removal prior to acid manufacturing. With either type, a complete precipitator installation consists of several individual subunits positioned both in series and in parallel to achieve the desired collection efficiency.

There are several important factors to be considered in the design and sizing of a precipitator to obtain a desired efficiency. These include the volume, temperature, and moisture content of the gas stream to be treated, and the resistivity, size distribution, and loading of the particulate to be collected. Of these, resistivity is the most important. Resistivity (the reciprocal of conductivity) is dependent on the chemical and physical properties of the particulate, and the temperature of the stream. Because of the presence of metal oxides, the resistivity of smelter dusts is relatively high at reduced temperatures (90 to 200°C) in the absence of natural conditioning agents such as moisture and $SO_3$. Too high a resistivity (greater than $10^{10}$ ohm-cm) may result in excessive sparking, which can seriously limit precipitator performance. Preconditioning with moisture and sulfuric acid has been applied to decrease particle resistivity prior to precipitation.[7]

3.1.1.5 Venturi Scrubbers. In a venturi scrubber, flue gases are passed through a venturi at high velocity, and the scrubbing liquid is introduced at the venturi throat under low static pressure where it is atomized and accelerated by the gas stream. Particles suspended in the gas stream are removed by impaction with the atomized liquid droplets. The wetted particles and entrained liquid are removed by a cyclone or other type of entrainment separator. The collection efficiency achieved, which can be in excess of 99 percent, is directly related to the total amount of energy expended in forcing the gases through the venturi and in atomizing and accelerating the scrubbing liquid. The expended energy is reflected in the pressure drop across the device, which may range from 2.5 kPa (10 in. $H_2O$) to over 20 kPa (80 in. $H_2O$). Typical throat velocities for venturi units range from 75 to 100 m/s (15,000 to 20,000 fpm) and liquid-to-gas ratios range from 0.4 to 2 liters/min per $m^3$/min (3 to 15 gpm/$10^3$ acfm).[8] It is important that the liquid-to-gas ratio be sufficiently high to guarantee complete wetting across the venturi cross-section.

The application of venturi scrubbers at primary copper smelters is limited to a few smelters where they are used to augment gas stream precleaning systems and to provide additional gas cooling prior to acid manufacturing.

3.1.1.6 Sulfuric Acid Plants. As noted previously, smelter offgases containing high concentrations of $SO_2$ (over 3.5 percent) are generally treated in single- or double-contact sulfuric acid plants for $SO_2$ removal. The presence of solid and gaseous contaminants, such as acid mist, entrained dust, and metal fumes in the treated smelter offgases can present serious difficulties in the operation of an acid plant. The major difficulties caused by these contaminants include the corrosion of heat exchanger tubes, plugging of catalytic beds, deactivation of the catalyst, and contamination of the product acid. As a result, rather extensive measures have to be taken to remove contaminants to ensure that their concentrations are reduced to tolerable levels prior to entering the acid plant.

Table 3-3 contains estimates of the maximum levels of impurities that can be tolerated in smelter offgases used for sulfuric acid manufacturing. The degree of catalyst deterioration experienced at these various impurity levels can be tolerated by shutting down the acid plant once per year to screen the catalyst and repair the equipment. Table 3-3 also contains the estimated upper level of impurities that can be removed by typical gas precleaning systems with prior removal of coarse dust. Although complete removal of contaminants, such as arsenic, from the offgases is not practical, over 99 percent removal is considered achievable.[10]

Both hot and cold gas cleaning devices are used. Generally, the offgases are initially treated in a hot electrostatic precipitator where the coarse particulate, which contains large amounts of metals, is removed. The gases exiting the precipitator are then scrubbed in one or more packed-bed or impingement-type scrubbers where, in addition to undergoing further particulate removal, the gases are humidified and cooled. The cooled gases then enter a series of electrostatic

Table 3-3. ESTIMATED APPROXIMATE MAXIMUM IMPURITY LIMITS FOR METALLURGICAL OFFGASES USED TO MANUFACTURE SULFURIC ACID[9]

| Substance | Impurity limits at inlet to acid plant, $(mg/Nm^3)^a$ | Impurity limits at inlet to gas purification system, $(mg/Nm^3)^{a,b}$ |
|---|---|---|
| Chlorides, as Cl | 1.2 | $125^c$ |
| Fluorides, as F | 0.25 | $25^d$ |
| Arsenic, as $As_2O_3$ | $1.2^e$ | 200 |
| Lead, as Pb | 1.2 | 200 |
| Mercury, as Hg | 0.25 | $2.5^f$ |
| Selenium, as Se | $50^e$ | 100 |
| Total solids | 1.2 | $1,000^g$ |
| $H_2SO_4$ mist, as 100% acid | 50 | |
| Water | | $400 \times 10^3$ |

[a] Basis: dry offgas stream containing 7 percent sulfur dioxide.

[b] For a typical gas purification system with prior coarse dust removal.

[c] Must be reduced to 6 $mg/Nm^3$ if stainless steel is used.

[d] Can be increased to 500 $mg/Nm^3$ if silica products in scrubbing towers are replaced by carbon; must be reduced if stainless steel is used.

[e] Can be objectionable in product acid.

[f] Can be increased to 5 to 12 $mg/Nm^3$ if lead ducts and precipitator bottoms are not used.

[g] Can be increased to 5,000 to 10,000 $mg/Nm^3$ if weak acid settling tanks are used.

mist precipitators where acid mist, fine particulates, and volatile metals are removed prior to entering the acid plant. If more elaborate cleaning is required, venturi-type scrubbers are used upstream of the cooling towers.

The basic steps in the contact process for the manufacture of sulfuric acid from sulfur dioxide-bearing gases are shown in Figure 3-2. As noted, the offgases are cooled and cleaned to remove particulates and volatile metals. Acid mist is removed in an electrostatic mist precipitator, and the gases are dried with 93 percent sulfuric acid. The dry gases then pass through a series of gas-to-gas heat exchangers to heat the offgases to the optimum temperatures for the catalytic conversion of sulfur dioxide ($SO_2$) to sulfur trioxide ($SO_3$). Single-contact acid plants use three or four stages of catalytic converters, whereas dual-contact plants use one, two, or three stages of catalyst before the first absorption tower. Since the conversion of sulfur dioxide to sulfur trioxide is exothermic, the converter outlet gases must be cooled before passing through the absorption tower. These outlet gases are passed countercurrent to the inlet gases in the heat exchangers mentioned above. The sulfur trioxide is then absorbed by 98 percent sulfuric acid in an absorption tower to yield the product. In a single-contact acid plant, the remaining gases are then treated to remove acid mist and spray, and then vented to the atmosphere.

In a dual-contact acid plant, the gases exhausted by the first absorption tower are passed through a second series of heat exchangers and catalytic converter stages to oxidize the sulfur dioxide remaining in the gases. Normally, this step employs one or two stages of catalyst. The gases then pass through a second absorption tower, where sulfur trioxide is absorbed by sulfuric acid as in the first absorption tower. The waste gases are then treated to remove acid mist and spray, and vented to the atmosphere.

3.1.1.7 Factors Affecting Performance. The alternative arsenic emission control techniques considered above share a common element. They all consist of cooling the gas stream to condense arsenic (provided a sufficient quantity of arsenic is present in the gas stream) and collecting the resultant fume in a high efficiency collection device.

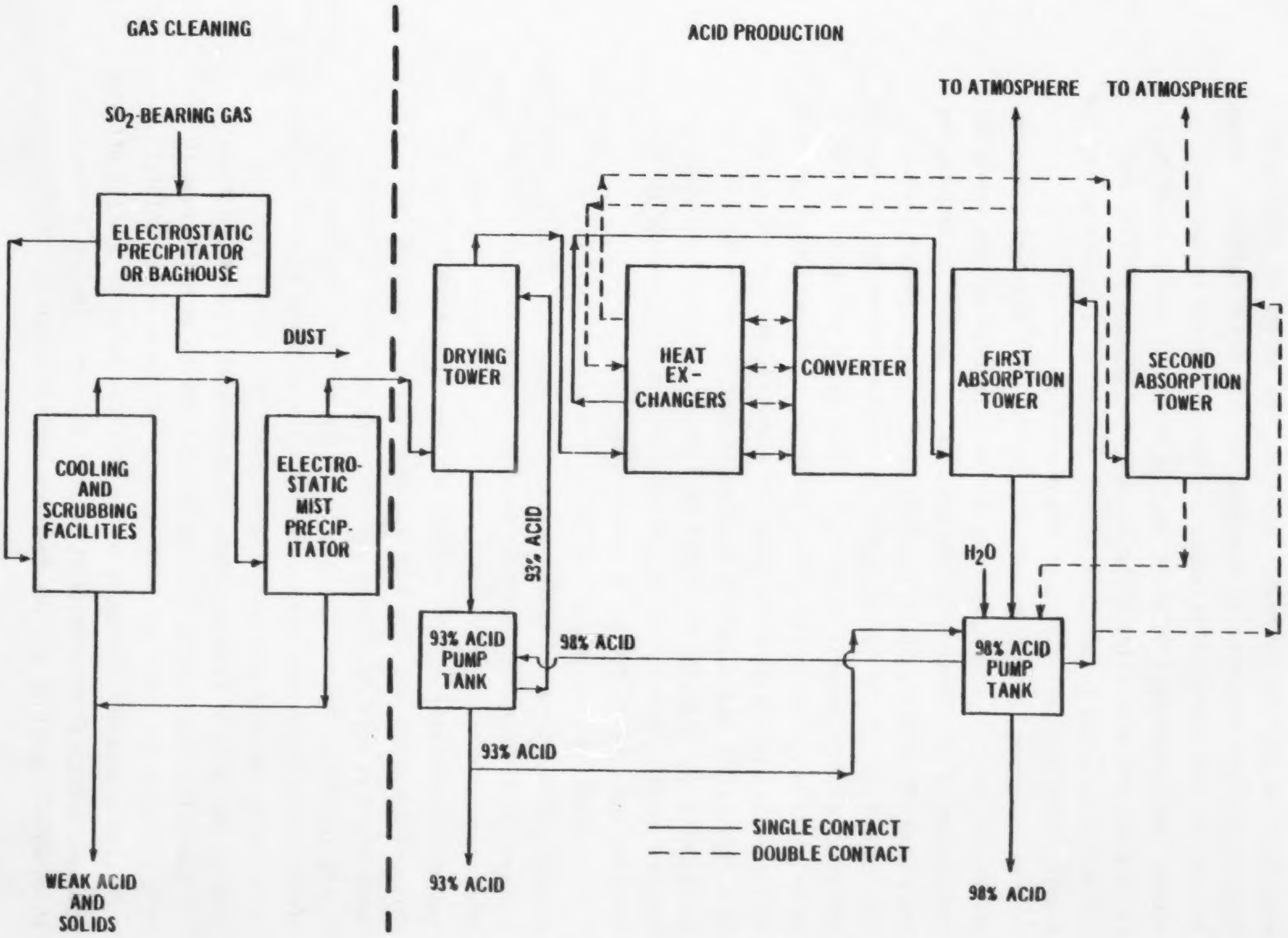

GAS CLEANING
ACID PRODUCTION
SO2-BEARING GAS
ELECTROSTATIC PRECIPITATOR OR BAGHOUSE
DUST
COOLING AND SCRUBBING FACILITIES
ELECTRO-STATIC MIST PRECIP-ITATOR
WEAK ACID AND SOLIDS
DRYING TOWER
93% ACID
93% ACID PUMP TANK
98% ACID
93% ACID
93% ACID
HEAT EX-CHANGERS
CONVERTER
FIRST ABSORPTION TOWER
SECOND ABSORPTION TOWER
TO ATMOSPHERE
TO ATMOSPHERE
H2O
98% ACID PUMP TANK
98% ACID
SINGLE CONTACT
DOUBLE CONTACT

Figure 3-2. Contact Sulfuric Acid Plant

Regardless of the control device employed, the amount of achievable arsenic emission reduction is dependent on four major factors. These include the concentration of arsenic trioxide present in the gas stream, the temperature to which the gas stream is cooled prior to collection, the allocation of sufficient time between cooling and collection to allow condensable arsenic compounds to condense, and the overall collection efficiency of the control device.

As discussed in Section 3.1.1.1, because of the volatile nature of arsenic ($As_2O_3$), the arsenic inlet concentration and the operating temperature of the control device are critical factors in determining its potential effectiveness in controlling the emission of arsenic. The temperature of the gas stream to be treated determines the maximum amount of arsenic present in the gas stream which can exist in the vapor state and, consequently, the quantity of arsenic which can exist in the solid state and be collected in a particulate control device. As a result, the gas stream to be treated must be cooled to the extent practicable and adequate time must be allowed to ensure that most of the condensable fraction of the arsenic present is condensed prior to entering the control device for collection.

Although cooling is basic to effective arsenic control, the presence of moisture and sulfur oxides in the smelter offgases introduces a lower temperature constraint below which further cooling cannot be tolerated without incurring severe operating problems. Because $SO_3$ is hygroscopic, it will absorb moisture at temperatures well above the moisture dew point and form highly corrosive sulfuric acid mist. The temperature at which acid mist formation occurs (acid dew point) is highly variable, depending on the $SO_3$ concentration and other gas stream characteristics. Continued operation of a dry control device at or below the acid dew point could result in a severe corrosion problem due to acid attack. Thus, it is generally recommended that the operating temperature of a dry control device be maintained 10 to 25°C (20 to 45°F) above the best estimate of the acid dew point.[11]

Little data are available on the acid dew point of smelter offgases. However, practical experience at several smelters, including the three ASARCO smelters and the Anaconda smelter prior to its closing, indicates that operating temperatures in the range of 100°C (212°F) to 110°C (230°F) are within tolerable limits.

The quantity or concentration of arsenic in the gas stream is very important in determining achievable arsenic emission reductions. To achieve any arsenic trioxide emission reduction by condensation, the quantity of arsenic trioxide in the gas stream must be sufficiently high so that the resultant arsenic trioxide concentration at the control device operating temperature exceeds the predicted saturation concentration. If the arsenic trioxide concentration at the control device operating temperature does not exceed the predicted saturation concentration, little or no emission reduction is achievable. Conversely, if the arsenic trioxide concentration greatly exceeds the predicted saturation concentration, arsenic emission reductions approaching the overall performance capability of the control device for particulate matter can be achieved.

The effect of the overall collection efficiency of the control device on achievable arsenic emission reduction is self-evident; the higher the efficiency for total particulate matter, the higher the efficiency for arsenic.

### 3.1.2 Fugitive Emission Sources and Controls

Fugitive emissions may be characterized as emissions which escape directly from the process area to the atmosphere rather than through a flue or exhaust system. They result from leakage in and around process equipment and from material handling and transfer operations.

Fugitive emissions from these sources are controlled by local ventilation (i.e., use of localized hoods or enclosures) or general ventilation techniques (i.e., building evacuation) to confine and capture emissions. Once captured, the emissions may be vented directly to a particulate control device or combined with process offgases prior to collection in a control device. As with process sources, some consideration must be given to cooling prior to collection. In most instances, however, cooling occurs naturally as a result of air dilution due to infiltration and, therefore, additional equipment is not required.

Besides the use of add-on controls, fugitive emissions from some sources may be minimized or eliminated by minor process changes and good operating and maintenance practices. A general discussion of these and other control alternatives is presented in the following subsections.

3.1.2.1. Local Ventilation. Local ventilation consists of using localized hoods or enclosures to confine the fugitive emissions at the source, and the use of induced air currents to entrain and capture the fugitive emissions and divert them into an exhaust opening.

The design of a local exhaust hood involves specifying its shape and dimensions, its position relative to the emission point, and its rate of air exhaust. The rate of exhaust is dependent on the air velocity required and on the size of the imaginary curving area of the hood. Air contaminants originating within this area are drawn directly into the exhaust opening. Methods for estimating the surface area of an exhaust hood can be found in standard references dealing with industrial ventilation. The capture velocity is the velocity of the air at the hood face or entry plane necessary to overcome opposing air currents and to capture the emission-laden air by causing it to flow into the exhaust hood.[11,12]

Hoods can generally be classified into three broad groups: enclosures, receiving hoods, and exterior hoods. Enclosures usually surround most of the point of emission, though sometimes one side may be partially or even completely open. Receiving hoods are those wherein the air contaminants are injected into the hoods. For example, the hood for a grinder is designed to be in the path of the high velocity dust particles. Exterior hoods must capture air contaminants that are generated from a point outside the hood itself, sometimes some distance away. A canopy hood is a good example of an exterior hood.

Exterior hoods are the most commonly used hoods and are by far the most difficult to design since they are the most sensitive to external conditions. For example, a hood that works well in a still atmosphere may be rendered completely ineffective by even a slight draft through the area.

For exhaust hoods to be effective, sufficient ventilation must be applied across the space between the source and the hood so that all emissions are entrained. This involves overcoming the cross currents of indoor air which could deflect the stream of fugitive emissions away from the hood. The ventilation rate for exhaust hoods applied to hot sources must also take into account the thermal draft that results from heat transfer from the source to the surrounding air. The exhaust rate should be as uniform as possible over the entire plane of the hood inlet. Various exhaust hood configurations are illustrated in Figure 3-3.

So-called "air curtains" can be used as complementary capture devices for local ventilation. Air curtains are basically air jets of suitable geometric configuration with sufficient momentum to resist the forces working against them and maintain their continuity across the opening they protect. For ventilation purposes, they are used as part of push-pull type systems, wherein the air jet or curtain is blown across the emission zone, forcing the emissions into an exhaust hood located opposite the air curtain slot. Typical examples of air curtain applications are shown in Figure 3-4. As indicated in Figure 3-4(a), hazardous fumes, e.g., vinyl chloride, can be contained by the use of air curtains. Dust control by air curtains is shown in Figures 3-4(b) and 3-4(c). A detailed discussion of air curtains is presented in Section 3.1.2.7.2.

The following typical fugitive emission sources at copper smelters can be controlled by local ventilation methods:

- Calcine transfer from roaster
- Matte tapping
- Slag tapping
- Converter

3.1.2.2 General Ventilation. This technique is required whenever it is not possible or expedient to use local exhaust hoods because of their handicapping operations, maintenance, or surveillance of the process or equipment; or when the local exhaust does not significantly reduce air requirements or has no worker exposure advantages.

General ventilation has historically taken the form of either natural air changes due to wind and density differences, or mechanically assisted air changes. Natural changes of air through a building in the absence of mechanical ventilation occur by the action of either of two forces, the play of wind through windows or other openings or the

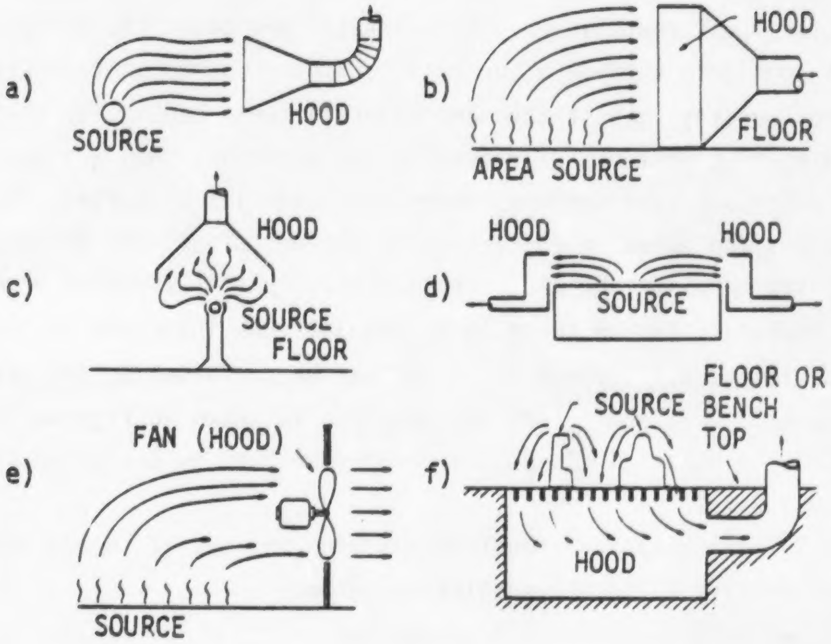

a)
SOURCE
HOOD
b)
HOOD
FLOOR
AREA SOURCE
c)
HOOD
SOURCE
FLOOR
d)
HOOD
HOOD
SOURCE
e)
FAN (HOOD)
SOURCE
f)
SOURCE
FLOOR OR BENCH TOP
HOOD

Figure 3-3. Types of Exhaust Hoods [12]

NOTE: In all cases, the source of contamination is beyond the boundaries of the hood structure; hence, control action is effected by inducing velocities in the adjacent space.

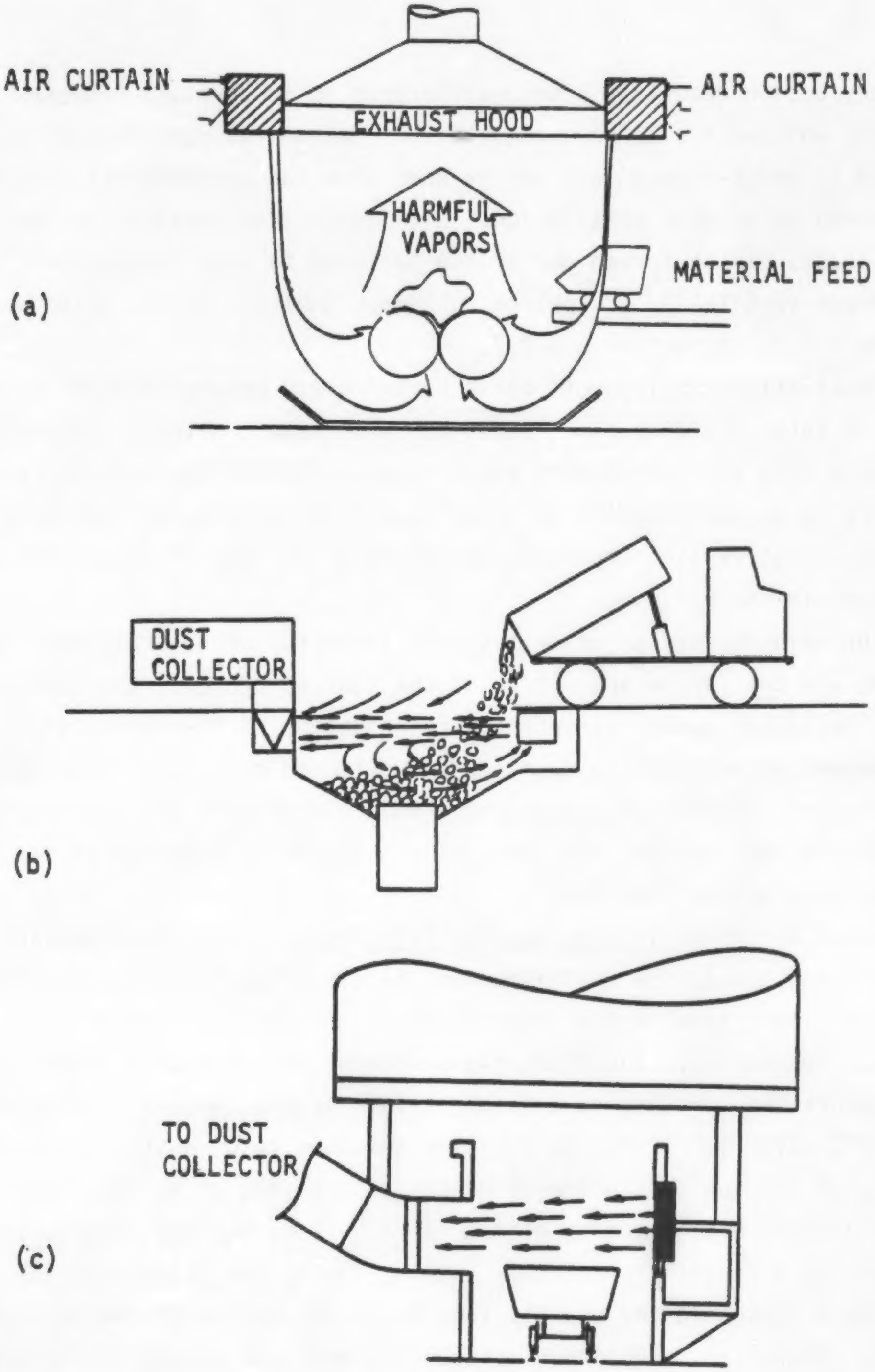

AIR CURTAIN
AIR CURTAIN
EXHAUST HOOD
HARMFUL VAPORS
MATERIAL FEED
(a)
DUST COLLECTOR
(b)
TO DUST COLLECTOR
(c)

Figure 3-4. Uses of Air Curtains[13,14]

NOTE: (a) for the control of hazardous fumes from vinyl chloride processes, (b) to control dust from a primary crusher operation, and (c) to control dust from loadout chutes.

buoyancy action resulting from a difference in temperature between outdoors and the inside of the building. Mechanical ventilation is induced by motor-driven fans and is used when the contaminants cannot be removed by natural ventilation. With mechanical ventilation the contaminants can be forced out of the building by roof evacuation. Mechanical ventilation as applied to copper smelters is discussed in greater detail in Section 3.1.2.7.

Ventilation requirements for a building are generally defined in terms of total building air changes per unit time. Although essential in determining the ventilation requirements, the air change rate, also referred to as the ventilation rate, is not the only factor which is important. It is also important to consider the rate of generation of emissions in the building.

The main factors on which successful control of general ventilation depends are the layout and siting of the sources of heat, the configuration of the building (number of spans, form and shape of the roof), and the arrangement of ventilation openings in walls and roof bays. The most satisfactory solutions are obtained when the architect and the engineer collaborate and consider the problems of natural ventilation at the design stage of the facility.

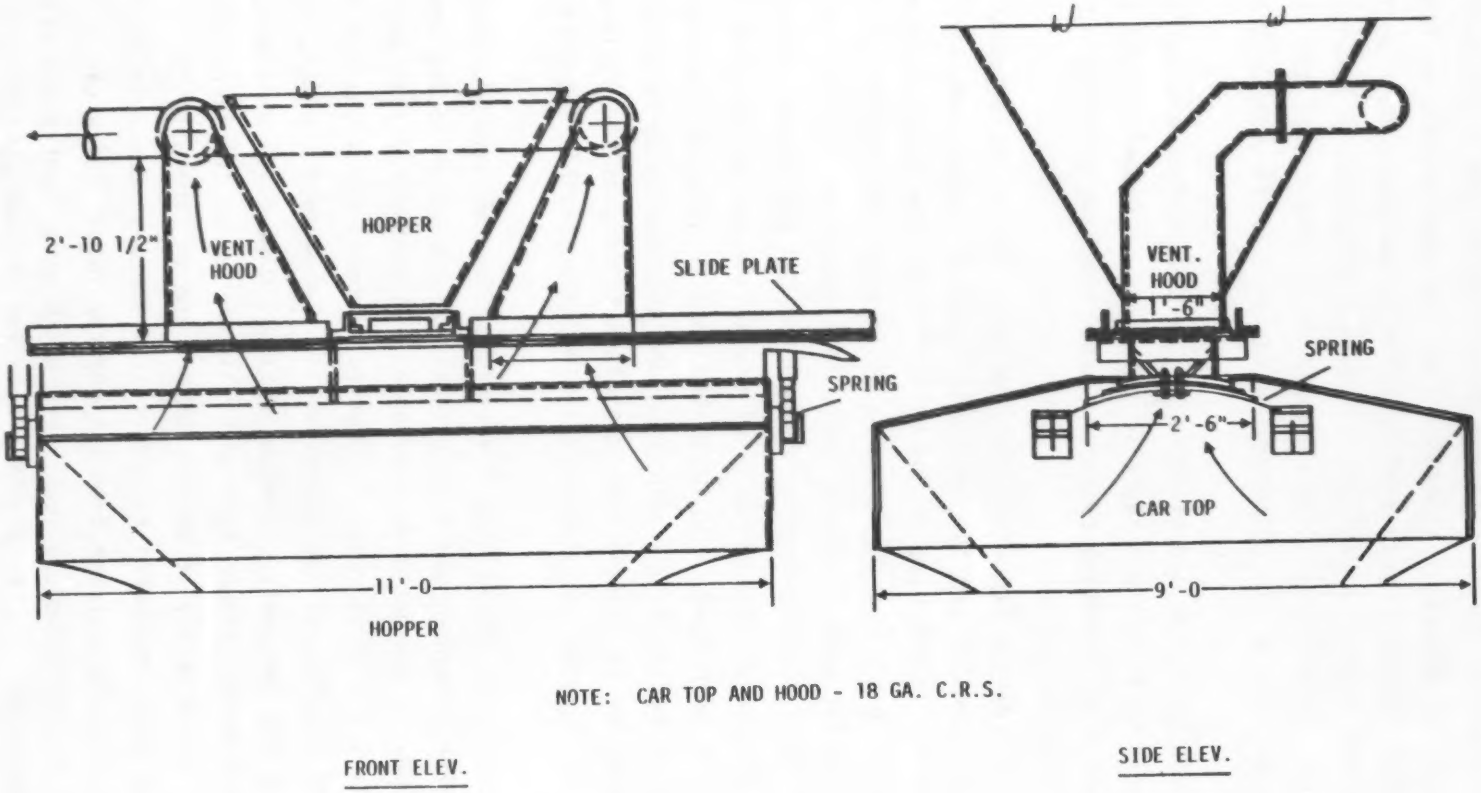
3.1.2.3 Collection Devices for Fugitive Sources. As discussed in Section 3.1.1.1, the most important factor affecting the controllability of arsenic emissions is the temperature of the uncontrolled offgas stream. Fortunately, fugitive offgas streams are generally lower in temperature than process streams and seldom have a temperature higher than 93°C (200°F). (Refer to fugitive emission test results in Appendix C.) Cooling of the fugitive offgases occurs as a result of air dilution due to ambient mixing and/or infiltration. Also, cooling is automatically achieved as a result of radiative cooling during the passage of the fugitive offgases in the usually long ductwork leading to the control device. Therefore, additional gas cooling prior to collection is not required for fugitive emissions.

The captured fugitive emissions are usually exhausted into existing process control systems. Only two domestic copper smelters use collection devices exclusively for the control of fugitive emissions. ASARCO-El Paso uses a baghouse to control fugitive emissions from the converter building evacuation system. Phelps Dodge-Douglas uses a baghouse to collect fugitive emissions generated from the roaster calcine discharge system. Both baghouses were tested by EPA to evaluate their performance. Results of the testing at ASARCO-El Paso are presented in Section 3.3.2.3, and in Tables C-25 through C-28 of Appendix C. Results of the Phelps Dodge-Douglas testing are presented in Tables C-53 through C-56 of Appendix C.

Although baghouses and electrostatic precipitators are currently used by smelters to collect fugitive emissions, scrubbers could also be used, but high operating costs and water handling problems make their use less desirable.

3.1.2.4 Calcine Transfer from Roaster. The multiple-hearth and fluidized-bed roaster are the two basic types of copper concentrate roasters used. In the case of the multi-hearth, the calcine hopper located at the bottom of the roaster, which drops the roaster calcine into the transfer vehicle (larry car), is a source of fugitive emissions. The use of close fitting hooding between the car and hopper outlet is difficult because of the necessity of moving the car to and from the roaster discharge hopper. Fluidized-bed roaster material transfer is carried out primarily by air conveying and secondarily by material dropout; transfer is usually well contained as long as the equipment is maintained in good condition, resulting in negligible fugitive emissions.

A schematic of the calcine transfer fugitive emission control system at ASARCO-Hayden, which is essentially identical to the one at ASARCO-Tacoma, is presented in Figure 3-5. A continuous flat apron strip nearly 0.6 m (2 ft) wide is mounted directly below the row of multi-hearth roasters at the hopper exits. Below each roaster, in the apron, there are two ports connected to a 0.5 m (1.5 ft) wide duct. A matching leaf spring-loaded flat apron is mounted on the larry car, which is driven directly below the roaster and in line with the matching holes on the apron connected to the roaster hopper. One hole is used for transferring the calcine from the roaster to the larry car. The other holes are connected to vent lines which go to vent hoods with their own individual draft fans (a single fan is used at ASARCO-Tacoma).



2'-10 1/2"
VENT. HOOD
HOPPER
SLIDE PLATE
SPRING
11'-0
HOPPER
VENT. HOOD
1'-6"
SPRING
2'-6"
CAR TOP
9'-0
NOTE: CAR TOP AND HOOD - 18 GA. C.R.S.
FRONT ELEV.
SIDE ELEV.

Figure 3-5. Spring-Loaded Car Top and Ventilation Hood, ASARCO -Hayden

Each draft fan has a capacity of approximately 142 $Nm^3$/min (5,000 scfm). The captured fugitives are then combined with the roaster process gases and treated in a baghouse.

In addition to the local hooding and ventilation applied directly at the calcine hopper discharge point, at ASARCO-Tacoma, the calcine hopper area has been enclosed to form a tunnel-like structure which is ventilated. The ventilated enclosure, coupled with the local hooding and ventilation applied at the actual calcine discharge point, is very effective in capturing fugitive emissions during calcine transfer operations. During visual observations, no fugitive emissions were observed escaping from the tunnel-like enclosure.[15]

3.1.2.5 Matte Tapping. Reverberatory furnaces may have up to four matte tap holes, two on each side of the furnace. Matte is tapped from one hole at a time and conveyed through troughs or launders into 4.9 to 9.2 $m^3$ (175 to 324 cubic feet) ladles. Typically, fewer than 30 taps per furnace are made each day, with each tapping operation taking 5 to 10 minutes. In electric furnaces there are generally four matte tap ports. Normally, only one matte tap port is in use at a time.

Copper matte from the furnace port travels through a launder which directs the flowing matte to a point where it can be collected in a large ladle. Emissions are observable from the point of the matte leaving the furnace to the point where it settles in the ladle.

Matte tap ports and launders in most smelting furnaces have hooding systems. Tap port exhaust hoods may be of any shape, as long as they are designed to be close to and cover as much of the emission area as possible. Launder hoods generally consist of covers mounted on the launder in sections to allow manual removal for launder cleaning.

Schematics of a matte tapping fugitive emission control system used at the ASARCO-Tacoma smelter are presented in Figures 3-6 and 3-7. The matte tap hood has a 1.2 by 1.2 m (4 by 4 ft) square cross-section and is located less than 0.9 m (3 ft) above the tap hole. The ducts connecting the matte tap exhaust hoods to the 1.2 m (4 ft) diameter main duct are approximately 0.6 m (2 ft) in diameter. During a tap, 283 $Nm^3$/min (10,000 scfm) is exhausted at the tap hole

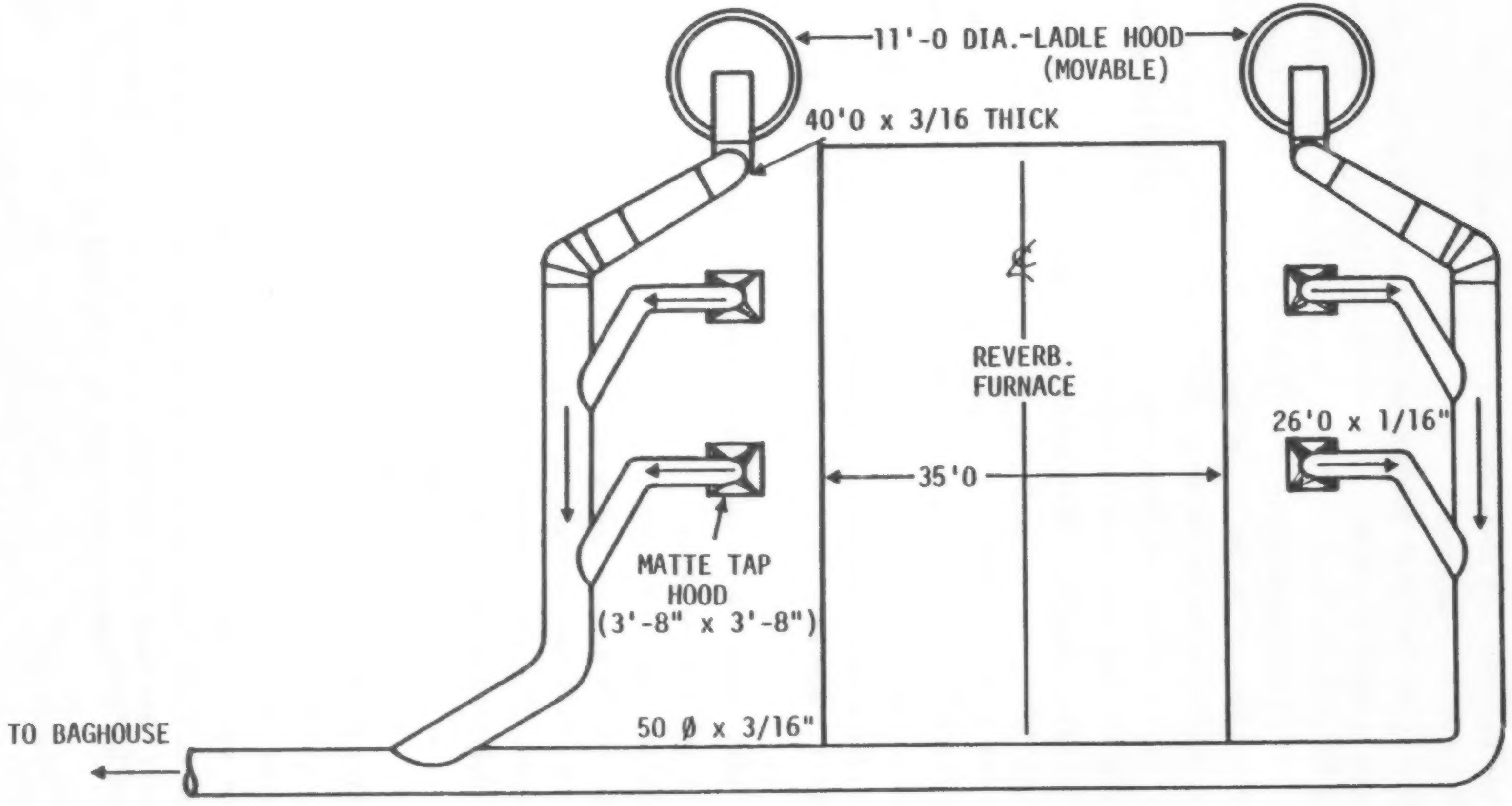

11'-0 DIA.-LADLE HOOD
(MOVABLE)
40'0 x 3/16 THICK
℄
REVERB.
FURNACE
35'0
26'0 x 1/16"
MATTE TAP
HOOD
(3'-8" x 3'-8")
50 Ø x 3/16"
TO BAGHOUSE

Figure 3-6. Matte Tapping Fugitive Control System (Plan View), ASARCO-Tacoma

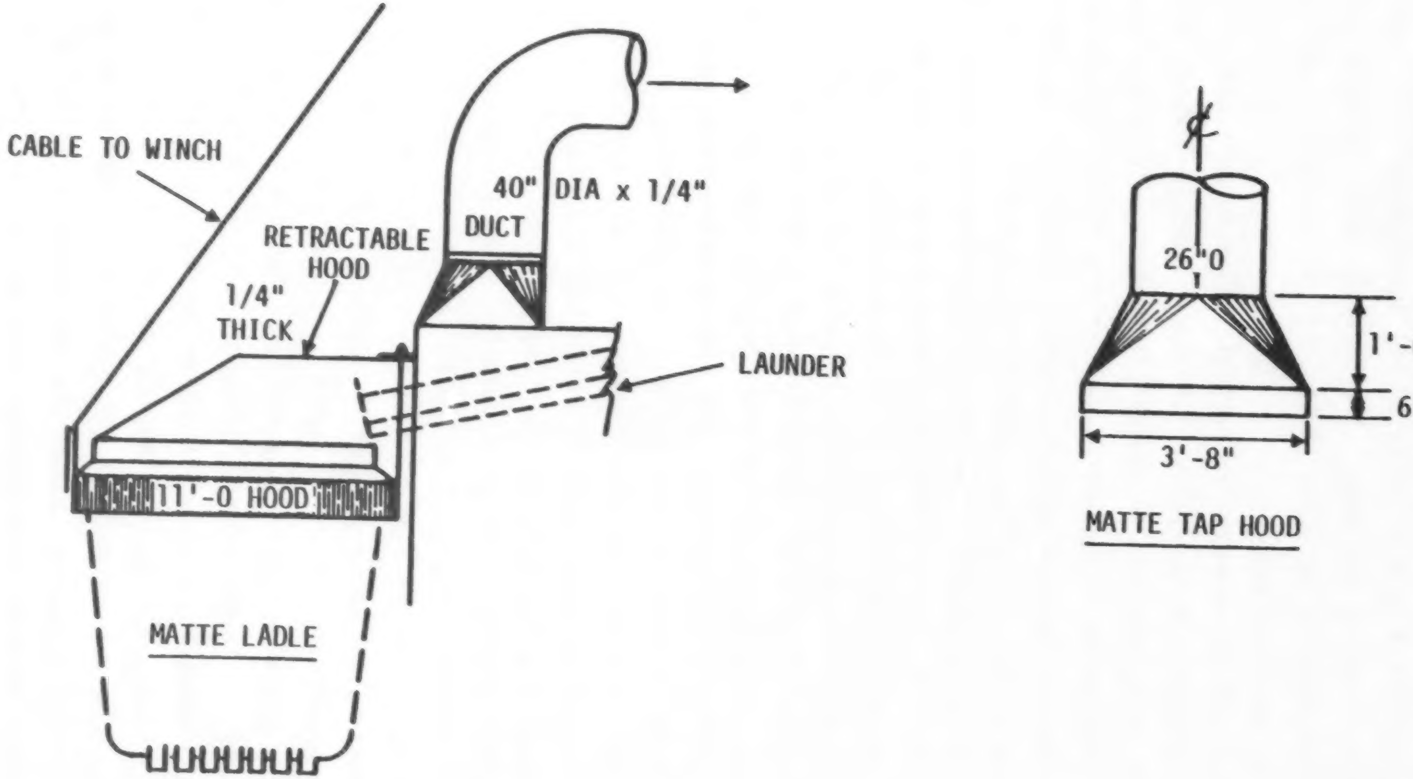

CABLE TO WINCH
RETRACTABLE HOOD
1/4" THICK
40" DIA x 1/4"
DUCT
LAUNDER
11'-0 HOOD
MATTE LADLE
26"0
1'-6"
6"
3'-8"
MATTE TAP HOOD

Figure 3-7. Matte Tapping and Ladle Hoods

hood and 850 $Nm^3$/min (30,000 scfm) is exhausted at the ladle. The covered launder is 3 to 6 m (10 to 20 ft) long, 0.6 m (2 ft) wide, and approximately 0.3 m (1 ft) deep. The launder covers (Figure 3-8) range in size from 0.6 to 1.5 m (2 to 5 ft) in length and have a semicircular cross-section.

A 3.4 m (11 ft) diameter retractable ladle hood is used to capture emissions generated at the ladle. The ladle hood is lowered over the ladle prior to tapping and is raised after the tap is completed.

From testing and observations made at the ASARCO-Tacoma smelter, the matte tapping fugitive emission capture system was observed to achieve greater than 90 percent capture efficiency.[15]

3.1.2.6 Slag Tapping. Generally, reverberatory furnaces have just one slag tap hole. Typically, slag is tapped an average of 30 times per day for approximately 10 to 20 minutes per tap. In electric furnaces there are usually two slag tap ports with one slag tap port in use at a time. The duration of a slag tap is normally 10 minutes. The slag flows from the tap port, down an inclined launder, and into one or more slag pots. The slag pots/ladles at various smelters range in capacity from 2.8 to 17 $m^3$ (100 to 600 cubic feet).

Fugitive emission capture techniques for slag tapping operations are very similar to those used for matte tapping. Local exhaust hoods are used over slag tap port areas. Slag launders are either partially or completely covered. Design volumes for exhaust hoods vary from one smelter to another, ranging from 566 to 850 $Nm^3$/min (20,000 to 30,000 scfm).

A schematic of the slag tapping fugitive emission control system used at the ASARCO-Tacoma smelter is shown in Figure 3-9. Slag tap hoods are pyramidical in shape, having a 1.2 m by 2.4 m (4 ft by 8 ft) rectangular cross-section. They are less than 0.9 m (3 feet) above the tap hole. A larger exhaust hood, 2.4 m by 4.3 m (8 ft by 14 ft), is situated above the slag pot transfer point. Each launder is covered with fixed hoods. During tapping, 142 $Nm^3$/min (5,000 scfm) is applied at the tap hole and 566 $Nm^3$/min (20,000 scfm) is applied at the slag pot. Emissions along the launder run are vented to either of the above hoods. As with matte tapping, a 90 percent capture efficiency should be achievable.

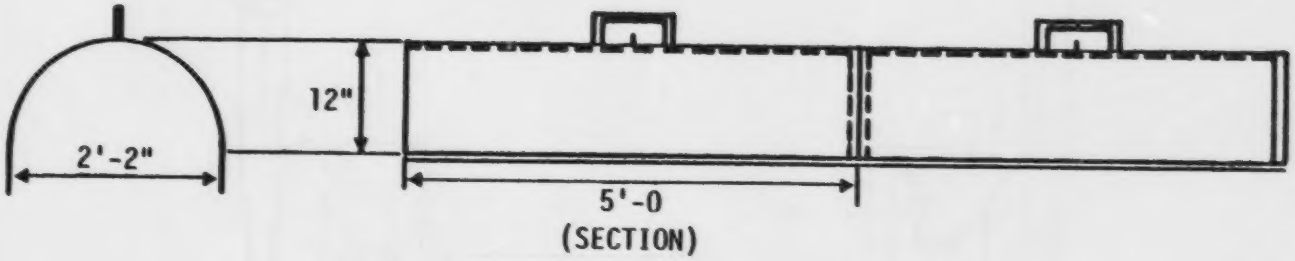
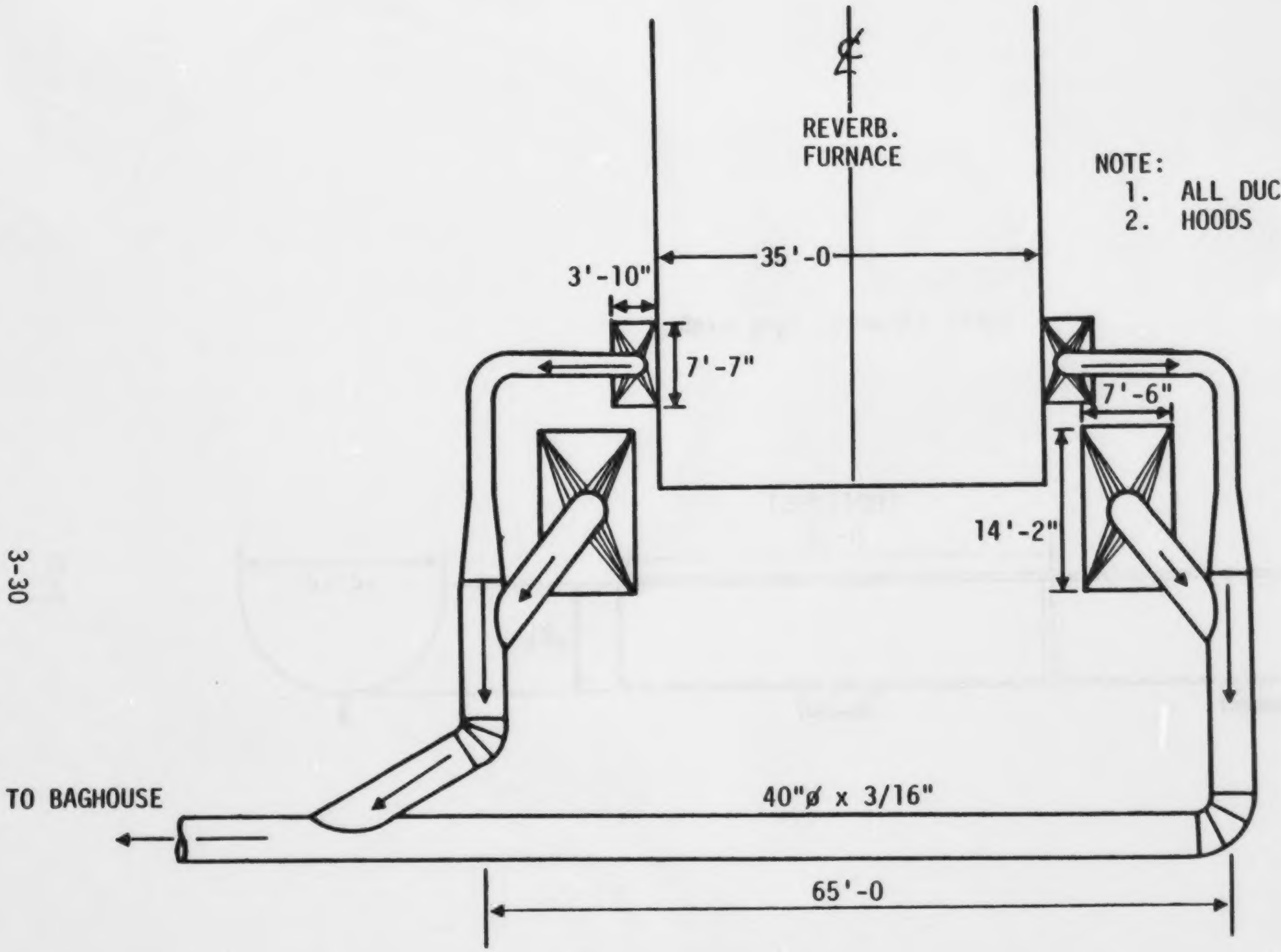

12"
2'-2"
5'-0
(SECTION)

Figure 3-8. Launder Cover


REVERB.
FURNACE
35'-0
3'-10"
7'-7"
7'-6"
14'-2"
TO BAGHOUSE
40"ø x 3/16"
65'-0

NOTE:
1. ALL DUCT - 3/16" THICK C.S.
2. HOODS - 3/16" THICK C.S.

Figure 3-9. Slag Tapping Fugitive Control System (Plan View), ASARCO-Tacoma

3.1.2.7 Converter Operations. Primary converter hoods capture process emissions during converter blowing periods, except for some emissions that escape due to primary hood leakage. However, during converter charging, skimming, or pouring, the mouth of the converter is no longer under the fixed primary hood, and extensive quantities of fugitive emissions escape capture by the primary hood.

Another source of fugitive emissions during converting occurs during the converter holding mode. This occurs under normal smelting operations when material, either slag or blister, cannot be immediately transferred from the converters to the ladles. During this period the generated emissions are not evacuated by the primary hood.

There are currently three basic alternative control techniques used to capture fugitive emissions during converting. They are: (1) secondary mechanical hoods; (2) air curtain secondary hoods; and (3) general ventilation/building evacuation.

3.1.2.7.1. Secondary mechanical hoods. In normal practice, primary converter hood systems are used only when the converters are in the blowing mode. At some smelters, secondary mechanical hoods are being used to capture emissions generated from the converter mouth during the other modes, such as charging, skimming, holding, and pouring. The flow rates handled by these hoods range from 700 to 2,400 $Nm^3$/min (25,000 to 85,000 scfm).

There are three major types of secondary mechanical hoods:

1. Fixed type - Attached to the primary or uptake hood, and currently used at Phelps Dodge-Ajo, Phelps Dodge-Hidalgo, Phelps Dodge-Morenci, and Kennecott-Utah.

2. Swing-away type - This type is being used at the Saganoseki smelter in Japan and consists of a swing-away type hood used as a deflector and a retractable type secondary hood just above it.

3. Mechanical type - This is a combination hood system utilizing a fixed hood, movable hood, gate hood, and swing-away hood. This system is a conceptual design system for converter fugitive emissions control.[16]

Domestic copper smelters are using the fixed type of secondary hooding to control fugitive emissions from converters. A typical converter secondary hood configuration of the fixed type is shown in Figure 3-10. These hoods are approximately 3 m (10 ft) long, 6.4 m (21 ft) wide, 1.7 m (5.5 ft) high and are affixed to the upper front side of the converter primary uptake hoods. Testing and observations made at existing copper smelters utilizing fixed converter hoods indicated that their effectiveness was only marginal.[17]

The swing-away type converter hood, sometimes called a deflector converter hood with a retractable secondary hood, is used at Japan's Saganoseki smelter. The hood was observed to be very effective during the blowing, slagging, and pouring of blister copper.[18] During charging of matte or rabbling, capture of the secondary emissions depended solely on the retractable hood portion of the device.

The converter mechanical secondary hood system shown in Figure 3-11 is a conceptual design developed by an EPA consultant.[19] This system uses a fixed hood in combination with a movable hood, a gate hood, and a swing hood. The elliptical fixed hood is made of steel and is attached to the primary uptake hood of the converter. Its opening is situated in a manner to avoid the hook of the overhead crane during converter operations. The upper end of the fixed hood is attached to the smoke plenum, which leads to dust bins on each side of the converter.

The elliptical movable hood (refer to Figure 3-11) is made of steel and fits over the fixed hood. It has its own track of movement. In the retracted position, this hood does not extend further, horizontally, than the fixed hood. In the extended position, it mates with the lip of the fixed hood to provide continuity of ducting for secondary emissions.

The elliptical gate hood fits under the fixed hood in the retracted position. In the extended position, the gate hood continues ducting of the secondary emissions to the movable and fixed hood.

The frustum-shaped swing hood can be rotated 180 degrees. The width at the top is the same as that of the mouth of the gate hood. This hood is made of steel with a castable refractory lining. Its pillar-mounting is motorized. In the retracted position it is clear of the aisle and slightly behind the converter.

Retrofitting the mechanical secondary hood system to an existing smelter could pose a difficult problem. The operational safeguards

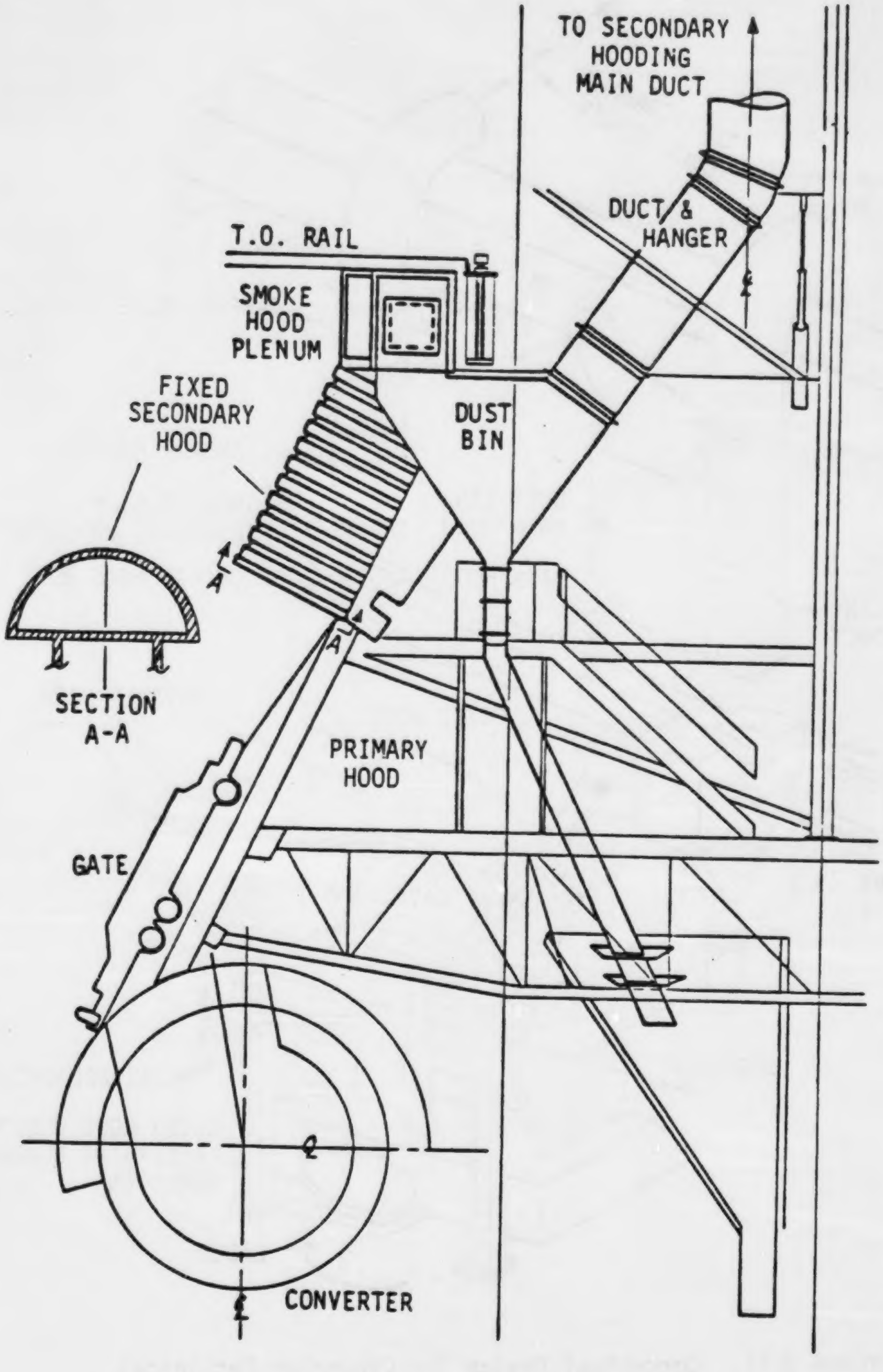

TO SECONDARY
HOODING
MAIN DUCT
DUCT &
HANGER
T.O. RAIL
SMOKE
HOOD
PLENUM
FIXED
SECONDARY
HOOD
DUST
BIN
A
A
SECTION
A-A
PRIMARY
HOOD
GATE
CONVERTER

Figure 3-10. Typical Converter Fixed Secondary Hood

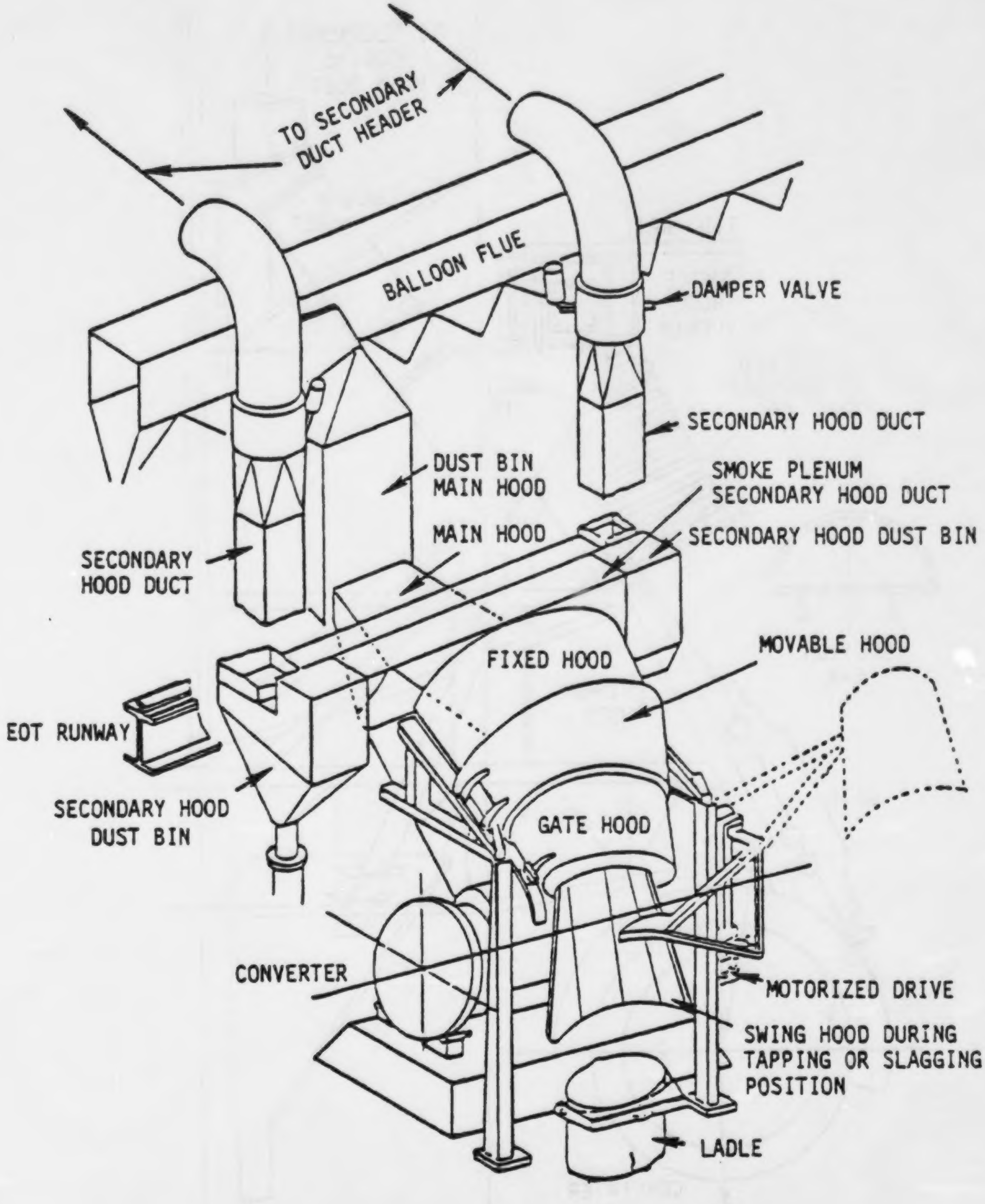

TO SECONDARY DUCT HEADER
BALLOON FLUE
DAMPER VALVE
SECONDARY HOOD DUCT
DUST BIN MAIN HOOD
SMOKE PLENUM SECONDARY HOOD DUCT
MAIN HOOD
SECONDARY HOOD DUST BIN
SECONDARY HOOD DUCT
MOVABLE HOOD
FIXED HOOD
EOT RUNWAY
SECONDARY HOOD DUST BIN
GATE HOOD
CONVERTER
MOTORIZED DRIVE
SWING HOOD DURING TAPPING OR SLAGGING POSITION
LADLE

Figure 3-11. Conceptual Design for Converter Mechanical Secondary Hood System

that would be required to operate it with a minimum number of breakdowns, delays, and damage would require close supervision and extensive effort. On the other hand, the maintenance and operation of the relatively inefficient fixed converter secondary hood system currently being used at domestic smelters and the swing-away converter secondary hood system, as used in Japan, are quite simple.

The efficiency of fixed converter secondary hoods is generally dependent upon the distance of the hood from the emission source and on the capture and face velocity created by the fan at the mouth of the fixed hood. Conversely, maintenance and durability are enhanced as the hood is moved farther from the converter mouth. The capture effectiveness of the fixed converter secondary hoods applied at the Phelps Dodge-Ajo and Phelps Dodge-Hidalgo smelters was judged to be low.[16]

The swing-away type of secondary converter hood again causes clearance problems with the crane hook and cables during collar pulling or matte additions. Also, rugged drive mechanisms are needed for the operation of the swing-away hood and added time is needed to complete pouring and skimming operations due to the time involved in the retraction of the swing-away hood to allow crane access to the ladle. However, good fugitive emission control is obtained during pouring, blowing, and slagging. During matte addition or rabbling, the capture efficiency is similar to that of a fixed hood.

Since the converter mechanical secondary hood system combines a fixed and a swing-away hood in combination with a movable hood, it should be more effective than either a fixed hood or swing-away hood used alone. The capture efficiency of the converter mechanical secondary hood system during charging, pouring, skimming, and blowing is estimated to be 40, 85, 70, and 85 percent, respectively.[20] Assuming a typical converter operation to consist of 80 percent blowing, 15 percent charging, and 5 percent skimming and pouring, an overall average capture efficiency for the mechanical secondary hood system is calculated to be approximately 80 percent.

3.1.2.7.2 Air curtain secondary hoods. Another method of controlling fugitive emissions from copper smelter converting operations involves the use of an air curtain system along with a secondary hood system. An air curtain secondary hood capture system has been installed on a domestic primary copper smelter (see below), and such systems are being used abroad[14,21,22,23] and in other U.S. industries.[24]

As discussed in Section 3.1.2.1, an air curtain is a suitably shaped air jet with sufficient momentum to resist the forces of fugitive gas streams working against it and to maintain its continuity across the opening it protects. Figure 3-12 shows a schematic diagram of an air curtain system controlling converter fugitive emissions. Consideration in air curtain design must be given to secondary or entrained flows which start forming as the air curtain jet stream leaves its slot or nozzle. As the entrained flows become fully mixed with the air curtain jet stream some distance from the nozzle, the hot or cold secondary flows are carried from one side of the air curtain jet stream to the other where they are ducted for suitable discharge. The greater the entrained flow, the greater the energy loss of the jet. To minimize energy loss, a relatively thick, slow moving jet stream with a large air volume is required. A basic rule in the design of air curtains is to use the thickest and lowest velocity air stream to be projected across the shortest dimension of the opening. Air curtain design methods are discussed in references 14, 25, and 26.

The air curtain system being used at the Onahama and Naoshima primary copper smelters in Japan is shown in Figure 3-13. The capture/shielding device includes two steel plate partitions, one on each side of the converter. The air jet is blown from a slot at the top of one of the plates across the opening to provide a sheet or curtain of air that prevents fugitive emissions from escaping. The other plate is equipped with an exhaust hood. The opening allows the crane cables to move into position above the converter mouth.

A propeller fan is used to push the air through an elongated slot on one side and a backward inclined fan provides suction on the opposite side to pull in both the fugitive gases and the push air. Captured gas passes through steel ductwork to a baghouse. The combined temperature of the converter fugitive gases with 100 percent of the push air

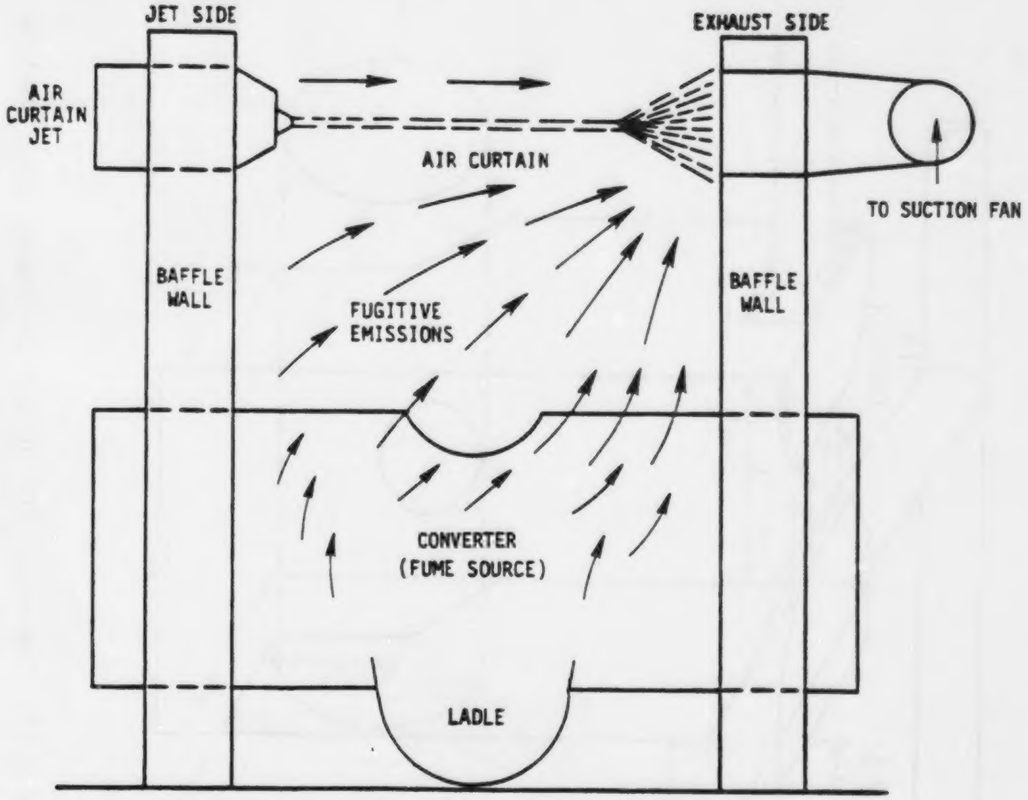

JET SIDE
EXHAUST SIDE
AIR CURTAIN JET
AIR CURTAIN
TO SUCTION FAN
BAFFLE WALL
FUGITIVE EMISSIONS
BAFFLE WALL
CONVERTER (FUME SOURCE)
LADLE

Figure 3-12. Converter Air Curtain Control System

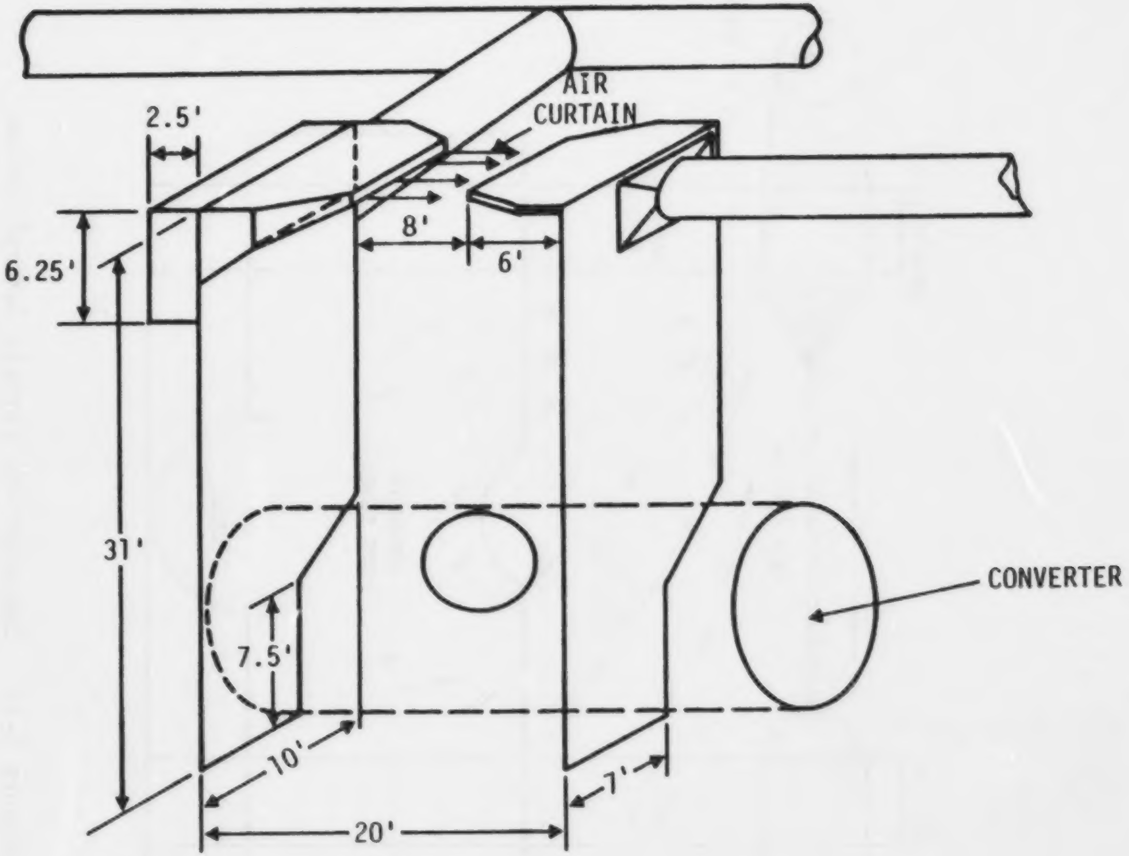

AIR
CURTAIN
2.5'
6.25'
8'
6'
31'
CONVERTER
7.5'
10'
7'
20'

Figure 3-13. Converter Air Curtain Secondary Hood, Onahama and Naoshima Smelters

entering the ductwork on the pull side of the air curtain is approximately 80°C (180°F), which makes gas cooling prior to the control device unnecessary.

The inlet air forming the air curtain above the converters at the Naoshima smelter has a flow rate of approximately 600 $Nm^3$/min (21,000 scfm). The exhaust hood on the opposite side pulls in approximately 1,000 $Nm^3$/min (35,000 scfm) of gas to the main system. The capacity of the total pull system at this smelter is three times this value, or 3,000 $Nm^3$/min (105,000 scfm), to accommodate the operation of three hoods at a time. According to Naoshima authorities, the collection efficiency of these hoods for fugitive emissions is approximately 90 percent.[27]

The Tamano copper smelter in Japan uses a differently designed air curtain system along with a fixed hood, which is essentially a total enclosure, for controlling fugitive emissions from each of its three converters (usually one converter is operated at a time.) A sketch of the air curtain system installed at the Tamano smelter is shown in Figure 3-14.[22] The enclosure has two front doors and a movable roof which is slightly inclined toward the front. The air curtain duct (slot hood) is located at the top of the enclosure level at a position to push air from one side of the converter to the other. Ambient air is supplied by a ground fan rated at 1,2000 $Nm^3$/min (41,000 scfm). Two ventilation points (offtakes) are located at the inside wall of the enclosure for capture of fugitive emissions; one is for dilute $SO_2$ gas and the other is for high concentration $SO_2$ gas. The inlet of the ductwork for high $SO_2$ gas is located near the converter mouth at a level below the other ductwork for dilute $SO_2$ fugitive gas. For convenience, the inlet of this ductwork is shown at the top of the roof in Figure 3-14.

Subjective evaluation of the air curtain secondary hood by visible observation at the Tamano smelter indicates the system to be at least 90 percent effective in controlling fugitive emissions.[22]

ASARCO's Tacoma facility recently installed a prototype air curtain secondary hooding system to control fugitive emissions from their No. 4 converter. In this system, walls are erected to enclose

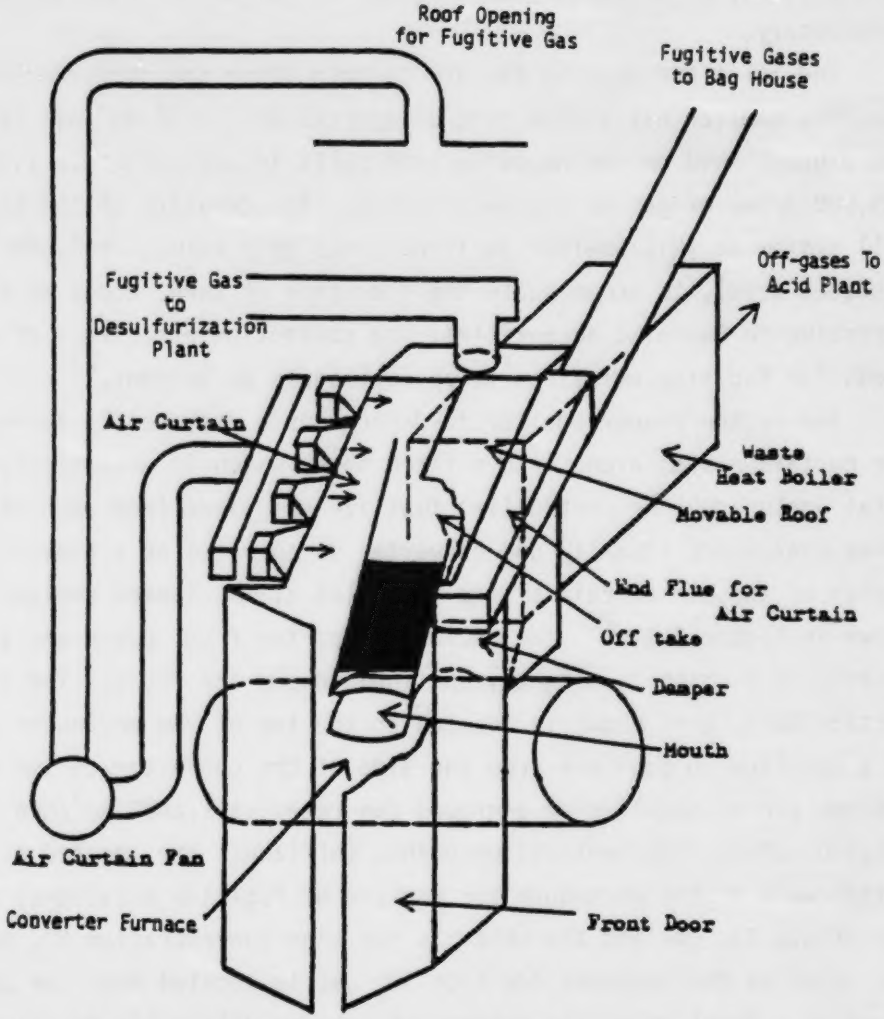

Roof Opening for Fugitive Gas
Fugitive Gases to Bag House
Off-gases To Acid Plant
Fugitive Gas to Desulfurization Plant
Air Curtain
Waste Heat Boiler
Movable Roof
End Flue for Air Curtain
Off take
Damper
Mouth
Air Curtain Fan
Converter Furnace
Front Door

Figure 3-14. Air Curtain System at the Tamano Smelter

the sides and the back of the area around the coverter mouth, with a portion of the enclosure back wall formed by the primary hood. Openings at the top and in the front of the enclosure allow for movement of the ladle and the overhead crane cables and block. The edges of the walls which contact the primary hood and the converter vessel are sealed to contain the emission plume.

When the converter is rolled out for charging or skimming, the gate on the primary hood is moved up and away from the converter mouth to provide clearance for the overhead crane and ladle. As noted previously, significant amounts of dust-laden fumes (fugitive emissions) escape the primary hood system during these modes of the converter cycle. Heavy fugitive emissions are generated during the roll-out and roll-in modes, when charging (including cold additions), slag skimming, and blister copper pouring are taking place. The heaviest emissions occur during the actual rolling out of the converter because the injection of blowing air continues during roll-out until the molten bath is below the tuyeres to prevent plugging by cooled material. During these periods of operation, the air curtain is utilized to capture most of the emissions which would otherwise escape the primary system.

Air volume control for the system is regulated automatically by dampers in the air curtain jet, the exhaust duct, and the induced draft fan. The dampers are manually set for a predetermined exhaust-side flow and, when placed in the automatic control position, are activated by movement of the primary hood and converter. When the primary hood is lifted and the converter is rolled out, the system switches to a high flow mode to control the heavy fugitive emissions generated during roll-out activities. At the completion of the converter roll-out operations, the converter is rolled in and the primary hood is lowered over the converter mouth. At this point, the system switches to a lower flow volume which is maintained during blowing and holding periods.

The gases that are captured by the air curtain system are treated in an ESP for particulate removal before being passed to the atmosphere through the main stack.

Design data for the ASARCO system are summarized in Table 3-4.

EPA conducted a program to evaluate the capture effectiveness of the ASARCO-Tacoma air curtain secondary hood system in January 1983.[29] This program included arsenic and total particulate sampling, a gas tracer study to determine the system's capture efficiency during specific modes of converter operation, and observations of visible emissions to assess the system's effectiveness. The results of this program are presented in Section 3.3.2.2.

3.1.2.7.3 Building evacuation. As noted previously, ventilation requirements for a building evacuation system are generally defined in terms of air changes per unit time. The rate of air change method estimates are based simply on room volume and do not consider the rate of evolution of the contaminant, the number of heat sources, or the natural draft due to building configuration. For example, a general ventilation installation designed by the rate of air change method can, under some conditions, actually cause the contaminant to be spread throughout the building, thus increasing the volume of dilution air required to maintain hygienic conditions. This occurs when the distribution of the ventilation air supply is poorly controlled. Uncontrolled air flowing into a building, due to negative pressure in the building or because of poorly designed air supply distributors, may not only cause recirculation of the contaminant, but also upset the local ventilation systems. It is, therefore, important that the amount of air, the location of its entry into the building, and its direction be controlled. For example, Figure 3-15 shows a controlled air supply which results in a convective flow from a heat source (such as a ladle of molten metal) rising to be exhausted through a roof ventilator. Figure 3-16 shows an uncontrolled air supply which results in a disrupted rising plume and recirculation of the contaminant throughout the building.

Natural air changes take place when hot air from the ground level heat sources rises due to its buoyancy. If, however, there exists a point within the building where the temperature of the surrounding air is equal to that of the rising column of hot air, buoyancy is lost. Therefore, natural air changes will take place only if the temperature

### Table 3-4. SUMMARY OF DESIGN DATA FOR THE ASARCO-TACOMA CONVERTER AIR CURTAIN SECONDARY HOOD SYSTEM[28]

| Mode of operation | Air Curtain Push Rate, Actual $m^3$/min | Main Offtake Evacuation Rate, Actual $m^3$/min |
|---|---|---|
| Matte charging | 510 (18,000 acfm) | 2,322 (82,000 acfm) |
| Blowing | -[a] | 1,700 (60,000 acfm) |
| Slag skimming | 510 (18,000 acfm) | 2,322 (82,000 acfm) |
| Holding | 510 (18,000 acfm) | 850 (30,000 acfm) |
| Worst conditions[b] | 1,020 (36,000 acfm) | 4,644 (164,000 acfm) |

[a] Air curtain will not be used during the blowing mode.

[b] Worst conditions would consist of either (1) two converters being charged simultaneously or (2) one converter being charged while another was being skimmed.

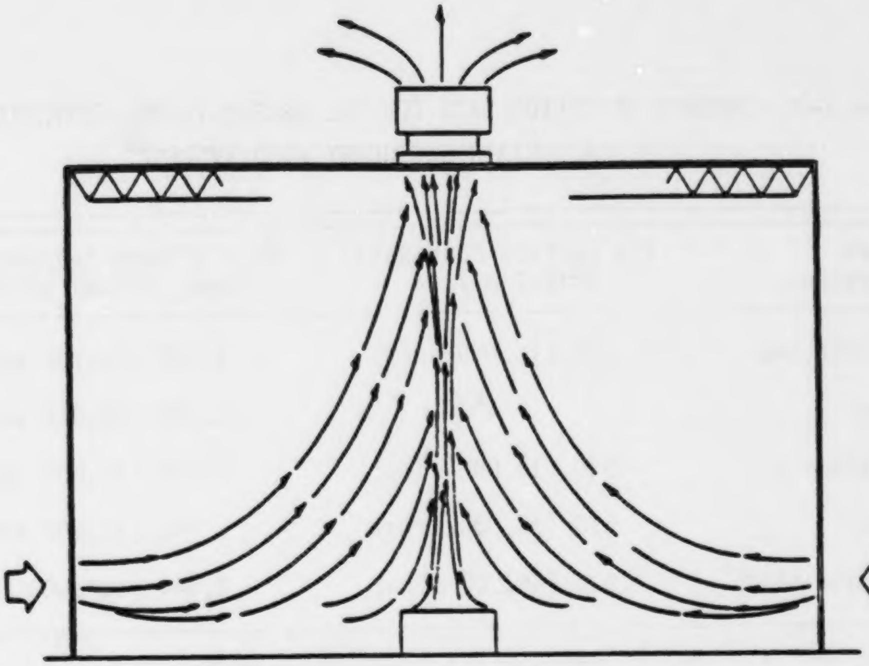


Figure 3-15. Controlled Airflow from a Heated Source[30]

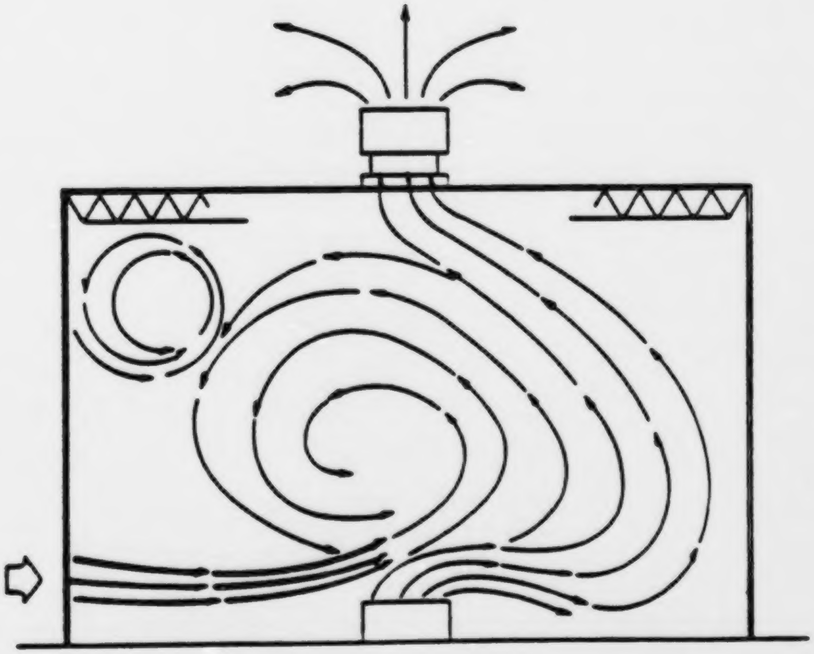


Figure 3-16. Uncontrolled Airflow from a Heated Source[30]

of the rising column of hot air is high enough to maintain the buoyancy of the column until it is discharged through the roof monitors. In most hot metal workshops this is, however, not the case. Hot air pools of some depth are formed under the building roofs. As a result, air entering the building will at times mix turbulently with pools of contaminated air and transport it downward to the occupied levels near the floor.

The concept of controlled ventilation is being employed at the ASARCO-El Paso smelter to capture emissions from the converter aisle. Several modifications (area isolation, vent location, air flow control, etc.) were made in order to implement this control measure at the facility. The present building evacuation rate is 16,800 Nm$^3$/min (600,000 scfm). This corresponds to an air change rate of 13 changes per hour. Although it is impossible to quantify, the building evacuation system at El Paso, when properly operated and maintained, is believed to be capable of achieving 95 percent capture. Particulate emissions contained in the building ventilation gases are controlled by a baghouse. EPA has performed tests on this baghouse; the results are provided in Section 3.3.2.3, and detailed test summaries are presented in Appendix C. Although the building evacuation system at ASARCO-El Paso should be capable of achieving 95 percent capture, the capture effectiveness actually being achieved is substantially less. Severe operating problems with this system have been encountered, which have led to unacceptable buildups of arsenic, lead, and heat in the building.[31] Worker exposure to airborne arsenic and lead has continuously exceeded the concentration limits set by the Occupational Safety and Health Administration (OSHA) (10 $\mu$g/m$^3$ for arsenic, 50 $\mu$g/m$^3$ for lead). In an attempt to alleviate the situation, the company has increased the openings in the building and are presently operating roof ventilators which discharge directly to the atmosphere. The roof ventilators were originally installed for emergency use only in the event of a power failure. The company is currently investigating the use of local ventilation methods (including converter secondary hoods) for use at its El Paso facility.

3.1.2.8 Anode Furnace. Anodes are fire refined directly from molten blister copper at all domestic copper smelters. Fire refining is carried out in rotary-type refining furnaces resembling Pierce-Smith converters or in small hearth furnaces. The rotary-type furnace is used at 14 of the 15 existing smelters. ASARCO-Tacoma is the only domestic smelter which employs hearth-type anode furnaces.

Fugitive arsenic emissions from anode furnaces contain little arsenic since most of the arsenic has been eliminated in earlier processing steps.

ASARCO-Tacoma is the only domestic copper smelter capturing arsenic emissions from anode furnaces. Emissions from the hearth-type anode furnaces at ASARCO-Tacoma are siphoned off and conveyed to an ESP for collection.

A swing-up type hood is used for the control of fugitive emissions from the rotary-type anode furnaces used in Japan. A schematic of such a hood is presented in Figure 3-17. This hood has a flexible duct approximately 0.7 m (2.25 ft) in diameter connected to an arc-shaped hood. The hood is approximately 4 m (13 ft) wide and 3 m (10 ft) long. The quantity of dilution air required to cool the gases is a function of the position of the hood above the anode furnace mouth. For an anode furnace temperature of 982°C (1,800°F), the quantity of dilution air required to reduce the temperature to 121°C (250°F) will be 198 $Nm^3$/min (7,000 scfm). The collection system is designed for 425 $Nm^3$/min (15,000 scfm) for one anode furnace under these conditions.

The emissions captured by the swing-up hood occur during the oxidizing and reducing blows. The emissions collected during these blows constitute the majority of the emissions generated. Some emissions escape during the charging of blister from the ladle and during the pouring of refined copper from the anode furnace, since the mouth of the furnace during these operations is not under the hood system; these emissions, however, are comparatively small.

3.1.2.9 Dust Transfer, Handling, and Conveying. Dust transfer, handling, and conveying practices vary from smelter to smelter. However, the dust transfer from control devices and smelter flues is common practice at all smelters.

Dust collected by an electrostatic precipitator drops into a storage hopper beneath the unit. A collecting conveyor, contained in dust-tight housing, transfers this dust to a storage bin from which

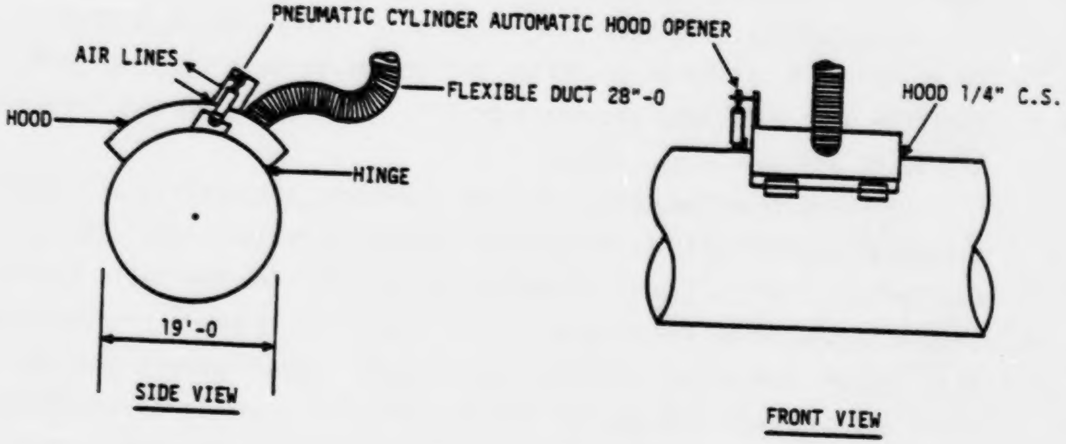

PNEUMATIC CYLINDER AUTOMATIC HOOD OPENER
AIR LINES
FLEXIBLE DUCT 28"-0
HOOD
HINGE
19'-0
SIDE VIEW
HOOD 1/4" C.S.
FRONT VIEW

Figure 3-17. Anode Furnace Movable Hood

dust may be withdrawn as desired. These bins are usually equipped with dust level indicators. Discharge from the storage hopper is usually through a dust-tight connection to surface transportation units. Normal practice is to distribute the dust on smelter feed beds where it is sprayed with water and covered by damp concentrates or precipitates and other furnace charge materials.

Dust collected by a baghouse drops into a holding hopper during the shaking cycle and is removed through a double gravity gate or rotary valve and into an enclosed collector conveyor (generally of the screw type), which in turn discharges into a closed storage bin. Depending on the nature of the collected dust and its metallurgical values, it is either conveyed to the charge mixing area or is discarded.

In scrubbers, the dust is removed as a slurry, which is then dewatered in a thickener or filter and/or passed to a settling pond. Fugitive dust emissions are not a problem in scrubber systems because dry dusts are not being handled.

Offgases from roasters, smelting furnaces, converters, and other pyrometallurgical units are conveyed through air-tight flues and chambers to their ultimate dispersal point. The coarser dusts tend to settle in the flues and chambers. All flues and chambers are equipped with hoppers spaced at intervals underneath. These hoppers provide surge capacity and storage for the settled dust until it is withdrawn. In the past, these hoppers were equipped with plain swing gates for discharge onto the ground, into railroad cars, or to other surface conveyance equipment. Naturally, there was some dust loss by this method. These hoppers are now equipped with discharge gates which usually automatically feed into screw or drag type collector conveyors for storage in a central receiving bin. The dust is then conveyed to receiving points. Flue dust, however, is a coarser material and may be dumped directly onto wet feed belts or conveyed to a desired location by pneumatic conveyors. Waste heat boilers and crossover flues present special problems because the dust from the smelting furnace is at high temperature and builds up in fusions and large accretions. These must be removed by manual means, water-bomb lances, soot blowers, fluxing, sand blasting, slugger guns, or a combination of methods. Due to the various sizes of materials dropping into the boiler ash pits, this material is handled and transported by conventional surface vehicle conveyance and is usually passed through a smelter flux crushing system.

## 3.2 SUMMARY OF EXISTING CONTROL

This section presents a description of air pollution control equipment applied at the ASARCO-Tacoma smelter for the control of process emissions and fugitive emissions.

### 3.2.1 Process Control Equipment at ASARCO-Tacoma

Offgases from the roasters, which average about 3,570 $m^3$/min (126,000 acfm) at 260°C (500°F), are combined with the exhaust gases from the ancillary calcine discharge fugitive emission control system [150 $Nm^3$/min (5,200 scfm)] and are treated in a baghouse for particulate removal. Prior to entering the baghouse, about 850 $m^3$/min (30,000 acfm) of tempering air is introduced to reduce the inlet temperature to less than 121°C (250°F). The baghouse consists of 17 compartments containing 120 bags in each. The bags are made of acrylic and measure 20 cm (8 inches) in diameter and 7.6 meters (25 feet) in length. The total baghouse filtering area is 9,950 $m^2$ (107,100 $ft^2$). The baghouse is designed to treat 5,665 $m^3$/min (200,000 acfm) of gas at an air-to-cloth ratio of about 0.6 $m^3$/min per $m^2$ (1.9 cfm/$ft^2$). Bag cleaning is performed by mechanical shakers. The clean baghouse exhaust is vented through a flue to the smelter main stack.

Process gases from the reverberatory furnace, which average about 1,415 $Nm^3$/min (50,000 scfm), pass through a pair of waste heat boilers where the gases are cooled to about 400°C (750°F). The exiting gases then pass through a large rectangular brick flue where additional cooling and gas stream conditioning take place by air infiltration and water and sulfuric acid sprays located in the flue. The resultant gas stream, about 5,100 actual $m^3$/min (215,000 acfm) at about 110°C (230°F), then enters the first of two electrostatic precipitators (in series) for particulate removal. The first precipitator is of the tube design and consists of 18 sections with a total collection area of about 6,619 $m^2$ (71,250 $ft^2$). Each section contains 84 pipes measuring 30 cm (12 inches) in diameter and 4.6 m (15 feet) in length. The second unit is of the plate-type design, consisting of seven parallel chambers, each with four fields in series, and has a total collection area of 7,710 $m^2$ (82,992 $ft^2$). The exiting gases, about 7,740 actual $m^3$/min (270,000 acfm) at 93°C (200°F), are discharged through a large flue to the smelter's main stack.

Offgases from converter blowing operations are captured by water-cooled hoods and passed through a series of multicyclones and a settling flue for coarse particulate removal prior to entering the gas cleaning circuits of either a liquid $SO_2$ plant or a single-contact sulfuric acid plant. The gas cleaning circuits for both plants are similar and consist of a water spray chamber, an electrostatic precipitator, scrubbers, and a mist precipitator, all being in series. The single-contact acid plant has a 182 Mg/day (200 ton per day) capacity at 5 percent $SO_2$ and is capable of processing 650 $Nm^3$/min (23,000 scfm) of converter gas. The liquid $SO_2$ plant processes up to 1,270 $Nm^3$/min (45,000 scfm) of the converter gases. The plant uses dimethylaniline (DMA) to absorb the $SO_2$ in the gas stream and uses steam stripping for regeneration. The 100 percent concentrated $SO_2$ gas stream produced is then liquefied by compression and the liquid $SO_2$ stored.

Offgases from the three hearth-type anode furnaces [1,400 $Nm^3$/min (50,000 scfm) at about 93°C (200°F)] are treated for particulate removal in a plate-type ESP before being discharged through the main stack.

Emissions from the arsenic plant at ASARCO-Tacoma are controlled by a baghouse. In addition, several smaller baghouses are used to minimize dust emissions from drag conveyors, charge hoppers, and storage bunkers. The main baghouse consists of five compartments containing 240 acrylic bags each. The baghouse treats about 1,200 $Nm^3$/min (42,000 scfm) of gases from the metallic arsenic plant, arsenic plant top flue, arsenic kitchens, and Godfrey furnaces. The air-to-cloth ratio is 1.2 $m^3$/min per $m^2$ (3.98 cfm/$ft^2$) at 1,700 $m^3$/min (60,000 acfm), at the design temperature of 93°C. Bag cleaning is performed by an air pulse system. The clean baghouse exhaust is vented to the smelter's main stack.

The estimated control efficiencies of the process emission control equipment at ASARCO-Tacoma are presented in Table 3-5. Performance testing of the process controls is discussed in Section 3.3.1.

Table 3-5. ESTIMATED CONTROL EFFICIENCIES OF EXISTING PROCESS EMISSION CONTROLS AT ASARCO-TACOMA

| Process emission source | Control device | Estimated current control efficiency |
|---|---|---|
| Multi-hearth roasters | Baghouse | 99 |
| Reverberatory furnace | ESP | 98 |
| Converters | Acid plant | 99 |
| Anode furances | ESP | 96 |
| Arsenic plant | Baghouse | 98 |

### 3.2.2 Fugitive Control Equipment at ASARCO-Tacoma

It should be noted that fugitive emission controls are not applied extensively throughout the copper smelting industry. Where applied, controls typically consist of using local ventilation techniques (i.e., hoods and enclosures) to confine and capture emissions. The captured emissions are typically discharged directly to the atmosphere through a stack. In some instances, however, captured fugitive emissions are treated for particulate removal in a control device either individually or combined with process offgases prior to treatment. A discussion follows of existing fugitive emission control systems in operation at ASARCO-Tacoma.

Fugitive emissions which could escape from the discharge of calcine from the roaster are confined and captured by close-fitting exhaust hoods located at the discharge point. The exhaust stream is vented into the main roaster flue by two 74 $Nm^3$/min (2,600 scfm) fans for treatment in a baghouse system (see Section 3.2.1). The sides of the larry car loading area are enclosed. Larry car covers are used while cars are in transit. Empty cars are parked under a ventilation connection.

Potential fugitive emissions during charging of the reverberatory furnace are minimized by the use of a manual control override, which simultaneously opens the furnace flue control damper and reduces the fuel supply to the furnace prior to each charge to prevent pressure surges in the furnace. Fugitive emissions generated during both matte and slag tapping operations are controlled by local ventilation techniques. Tapping ports, launders, and launder-to-ladle/pot transfer points are hooded and ventilated. Ventilation for the matte tapping system totals about 700 $m^3$/min (25,000 acfm), while the ventilation for the slag tapping system totals about 600 $m^3$/min (21,000 acfm). Captured emissions from both systems are combined with anode furnace offgases and treated in an electrostatic precipitator (ESP) for particulate removal prior to being discharged through the main stack.

As mentioned earlier, several small baghouses are utilized to control potential fugitive dust emissions from the arsenic production building. These baghouses collect dust generated during material transfer operations, such as charging of the Godfrey roasting furnaces. Housekeeping practices, such as manual vacuuming and wet cleaning of work areas, are also a primary means used to minimize dust entrainment into the air during operations at the arsenic plant.

Table 3-6 presents a summary of the estimated fugitive emission control efficiencies currently being achieved at ASARCO's Tacoma smelter. A discussion of fugitive controls at this smelter is contained in Section 3.3.2.

Table 3-6. ESTIMATED CONTROL EFFICIENCIES OF EXISTING FUGITIVE EMISSION CONTROLS AT ASARCO-TACOMA

| Emission source | Estimated current control efficiency, percent | | |
|---|---|---|---|
| | capture | collection | overall |
| Calcine transfer | 90 | 99.8 | 89.8 |
| Matte tap | 90 | 96 | 86.4 |
| Slag tap | 90 | 96 | 86.4 |
| Converters | 0 | 0 | 0 |
| Anode furnace | 0 | 0 | 0 |
| Arsenic building | a | a | 90 |
| Miscellaneous | a | a | 90 |

[a] See Table 2-11.

## 3.3 PERFORMANCE CAPABILITIES OF ALTERNATIVE CONTROL TECHNIQUES FOR ARSENIC AND TOTAL PARTICULATE EMISSIONS

In order to evaluate the emission control capabilities of alternative control techniques for arsenic emissions and total particulate, EPA conducted emission measurements and made visual observations at several smelters including the ASARCO-Tacoma smelter. This section discusses the results of these emission measurements and presents conclusions regarding the performance capability of each of the control techniques evaluated for process and fugitive emission sources.

### 3.3.1 Process Control Systems

3.3.1.1 Baghouses. Tests were performed at the ASARCO-Tacoma and Anaconda* smelters to evaluate the performance of baghouses in controlling arsenic emissions. Tests were also performed at the Anaconda smelter to evaluate the performance of the baghouse in controlling total particulate emissions. Two baghouses were sampled at ASARCO-Tacoma and one at Anaconda.

3.3.1.1.1 Baghouses (ASARCO-Tacoma). Simultaneous inlet and outlet arsenic emission measurements were performed by EPA across the baghouse serving the multi-hearth roasters at ASARCO-Tacoma. A description of the control system was given in Section 3.2.1.

Sampling at the inlet was performed in the duct carrying emissions from the roasting process in four 10 cm (4 in.) ports on top of the flue. The gas leaving the roaster baghouse was sampled approximately 305 m (1,000 ft) downstream from the baghouse. Three tests were performed at each location. The results are summarized in Table 3-7.[32]

As indicated, the average arsenic inlet concentration and corresponding mass rate were 288 $mg/Nm^3$ (0.126 gr/dscf) and 87 kg/hr (191 lb/hr), respectively. The outlet arsenic loading was 0.9 $mg/Nm^3$ (0.0004 gr/dscf), and the mass rate was 0.3 kg/hr (0.6 lb/hr). The average arsenic removal efficiency for this unit, as indicated by these results, was 99.7 percent.

Simultaneous inlet and outlet arsenic emission measurements were also conducted at the baghouse used to treat a combination of offgases from the arsenic production facilities at ASARCO-Tacoma. These facilities

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*No longer in operation.

## Table 3-7. ARSENIC PERFORMANCE DATA FOR THE ROASTER BAGHOUSE AT ASARCO-TACOMA

| Sample run | Arsenic measurements[a] | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| | °C (°F) | mg/Nm$^3$ (gr/dscf) | kg/hr (lb/hr) | °C(°F) | mg/Nm$^3$ (gr/dscf) | kg/hr (lb/hr) | |
| 1 | 94 (201) | 314.3 (0.138) | 92.7 (204) | 88 (191) | 0.6 (0.0003) | 0.2 (0.4) | 99.8 |
| 2 | 85 (185) | 295.8 (0.130) | 88.1 (194) | 87 (189) | 0.7 (0.0003) | 0.2 (0.4) | 99.8 |
| 3 | 95 (203) | 254.2 (0.111) | 79.5 (175) | 82 (180) | 1.5 (0.0007) | 0.5 (1.0) | 99.4 |
| Avg. | 91 (197) | 288.1 (0.126) | 86.8 (191) | 86 (187) | 0.9 (0.0004) | 0.3 (0.6) | 99.7 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

include six Godfrey roasters, arsenic kitchens, and a metallic arsenic plant. The baghouse, which has been replaced with a new baghouse, consisted of five compartments, each having 288 homopolymer acrylic bags. The bags measured 320 cm (126 in.) in length and approximately 13 cm (5 in.) in diameter. The total filtering area was 1,860 $m^2$ (20,000 $ft^2$). It was designed to effectively treat 850 $Nm^3$/min (30,000 scfm) at an air-to-cloth ratio of 0.63 $m^3$/min per $m^2$ (2.06 cfm/$ft^2$). Bag cleaning was performed by mechanical shakers. The outlet sampling was done approximately 150 m (500 ft) downstream of the baghouse. The test results obtained are summarized in Table 3-8.[33]

As indicated, the average arsenic inlet concentrations and corresponding mass rate at the baghouse were 2,941 mg/$Nm^3$ (1.28 gr/dscf) and 76.3 kg/hr (168 lb/hr), respectively. The outlet arsenic loading was 60.6 mg/$Nm^3$ (0.026 gr/dscf), and the mass rate was 3.3 kg/hr (7 lb/hr). The average arsenic removal efficiency for this unit as indicated by these results was 95.7 percent.

3.3.1.1.2 Spray chamber/baghouse (Anaconda). Inlet and outlet measurements were made at the spray chamber/baghouse separately for arsenic and total particulate emissions. The sampling locations included the two inlet plenums to the spray chamber and the baghouse exhaust duct. Three samples each for arsenic and particulate were obtained at the inlet and outlet. The results of the arsenic emission measurements and particulate emission measurements are summarized in Tables 3-9 and 3-10, respectively.[34]

When the tests were conducted, the smelter had the following process emission control configuration. Smelting facilities at the Anaconda smelter consisted of a fluid-bed roaster, a single electric furnace, and six converters. Except for a portion of the electric furnace and converter offgases [about 2,070 $Nm^3$/min (73,000 scfm)], which were diverted to an acid plant for $SO_2$ removal, process gases from all three major smelting operations were combined into a main flue duct and transported to a baghouse system for particulate removal prior to being discharged to the atmosphere through the main stack.

Table 3-8. ARSENIC PERFORMANCE DATA FOR THE ARSENIC PLANT BAGHOUSE AT ASARCO-TACOMA

| Sample run | Arsenic Emissions[a] | | | | | | | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Inlet | | | | | | | | Outlet | | | | |
| | Arsenic kitchen | | | Metallic arsenic | | | Total | | | | | | |
| | °C | mg/Nm$^3$ | kg/hr | °C | mg/Nm$^3$ | kg/hr | mg/Nm$^3$ | kg/hr | °C | mg/Nm$^3$ | kg/hr | | |
| 1 | 56 | 1,717 | 42.4 | 117 | 1,931 | 42.6 | 3,648 | 85.0 | 83 | 69.8 | 3.5 | | 95.9 |
| 2[b] | 58 | 1,518 | 42.8 | 112 | 1 | 0.04 | 1,519 | 42.8 | 72 | 15.8 | 1.0 | | 97.7 |
| 3 | 60 | 1,532 | 45.1 | 97 | 2,125 | 55.9 | 3,657 | 101.0 | 71 | 96.1 | 5.5 | | 94.5 |
| Avg. | 58 | 1,589 | 43.4 | 109 | 1,352 | 32.9 | 2,941 | 76.3 | 76 | 60.6 | 3.3 | | 95.7 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

[b] During this sample run the metallic arsenic process may not have been operating.

## Table 3-9. ARSENIC PERFORMANCE DATA FOR SPRAY CHAMBER/BAGHOUSE AT THE ANACONDA-ANACONDA SMELTER

| Sample run | Arsenic Emissions[a] | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| | °C | mg/Nm$^3$ | kg/hr | °C | mg/Nm$^3$ | kg/hr | |
| 1 | 274 | 1,071 | 276.7 | 99 | 7.1 | 1.8 | 99.3 |
| 2 | 269 | 895 | 236.8 | 102 | 9.8 | 2.5 | 98.9 |
| 3 | 244 | 687 | 186.1 | 101 | 12.6 | 3.4 | 98.2 |
| Avg. | 263 | 885 | 232.3 | 101 | 9.8 | 2.6 | 98.9 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

## Table 3-10. PARTICULATE PERFORMANCE DATA FOR SPRAY CHAMBER/BAGHOUSE AT THE ANACONDA-ANACONDA SMELTER

| Sample run | Particulate Emissions[b] | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| | °C | g/Nm$^3$ | kg/hr | °C | g/Nm$^3$ | kg/hr | |
| 1 | 281 | 14.76 | 4,071 | 103 | 0.05 | 14.6 | 99.6 |
| 2 | 288 | 13.57 | 3,736 | 103 | 0.04 | 10.0 | 99.7 |
| 3 | 302 | 14.08 | 3,860 | 101 | 0.05 | 14.7 | 99.6 |
| Avg. | 290 | 14.14 | 3,890 | 102 | 0.05 | 13.1 | 99.7 |

[b] Concentration and mass rate data are based on measurements on the probe, cyclone, and filter catch (front half).

The baghouse system consists of a baghouse preceded by two parallel spray chambers, which effectively reduce the temperature of the inlet gas stream from about 315°C (600°F) to less than 104°C (220°F) prior to entering the baghouse. The baghouse consists of 18 compartments, each equipped with 240 Orlon bags, and has a net collection area of 29,800 $m^2$ (32,080 $ft^2$). It is designed to effectively treat 5,660 $Nm^3$/min (200,000 scfm) at an air-to-cloth ratio of 0.25 $m^3$/min per $m^2$ (0.80 cfm/$ft^2$). Bag cleaning is performed by mechanical shakers.

As indicated in Table 3-9, the arsenic inlet concentration and corresponding mass rate were extremely high, averaging 885 mg/$Nm^3$ (0.3867 gr/dscf) and 226.5 kg/hr (498.3 lb/hr), respectively. The outlet arsenic loading was 9.8 mg/$Nm^3$ (0.0043 gr/dscf), and the mass rate was 2.6 kg/hr (5.8 lb/hr). The average arsenic removal efficiency for this unit, as indicated by these results, was 98.9 percent.

Corresponding data in Table 3-10 indicate that the inlet particulate concentration and mass rate averaged 14.14 g/$Nm^3$ (6.2 gr/dscf) and 3,890 kg/hr (8,558 lb/hr), respectively. The outlet particulate loading was 50 mg/$Nm^3$ (0.022 gr/dscf), and the mass rate was 13.1 kg/hr (28.9 lb/hr). The average particulate removal efficiency for this unit was 99.7 percent.

3.3.1.2 Electrostatic Precipitators. Arsenic emission measurements were conducted on the electrostatic precipitator (ESP) units at the ASARCO-Tacoma and ASARCO-El Paso smelters to evaluate the performance of ESP's operated at reduced temperatures. In addition, the ASARCO-El Paso ESP was also tested for particulate matter.

3.3.1.2.1 Electrostatic precipitator (ASARCO-Tacoma). Arsenic emission measurements were conducted by EPA on the outlet of the ESP which treats offgases from the reverberatory furnace. Prior to entering the precipitator, the gases pass through a pair of waste heat boilers where the temperature of the gases is reduced to 400°C (750°F). The exiting gases then pass through a large rectangular brick flue where additional cooling and gas stream conditioning are provided by air infiltration and water and sulfuric acid sprays located in the flue. The resultant gas stream, about 6,100 actual $m^3$/min (215,000 acfm) at 132°C (270°F), then enters the first of two electrostatic precipitators in series for particulate removal. The first precipitator is a tube or pipe design consisting of 18 sections with a total collection area of about 6,619 $m^2$ (71,250 $ft^2$). Each section contains 84 pipes measuring 30 cm (12 inches) in diameter and 4.6 m (15 feet) in length. The second unit is of a plate-type design, consisting of seven parallel chambers each with four fields in series and with a total collection area of 7,710 $m^2$ (82,992 $ft^2$). The exiting gases, about 7,740 actual $m^3$/min (270,000 acfm) at 90°C (200°F), are discharged through a large flue to the smelter main stack.

Since the configuration of the inlet duct was such that it was not possible to sample, only outlet arsenic emission measurements were made by EPA at the electrostatic precipitator. The outlet sampling was performed in the duct 23 m (75 ft) downstream of the main stack. Three sample runs were made. A summary of the results obtained at the outlet is presented in Table 3-11.[35]

Table 3-11. ARSENIC EMISSIONS AT OUTLET OF REVERBERATORY FURNACE ELECTROSTATIC PRECIPITATOR AT ASARCO-TACOMA

| | | Arsenic Emissions[a] | |
|---|---|---|---|
| Sample run | °C (°F) | mg/$Nm^3$ (gr/dscf) | kg/hr (lb/hr) |
| 1 | 105 (220) | 38.1 (0.016) | 28.7 (63.1) |
| 2 | 101 (214) | 21.0 (0.009) | 11.7 (25.8) |
| 3 | 87 (188) | 9.6 (0.004) | 7.1 (15.6) |
| Avg. | 97 (207) | 22.9 (0.010) | 17.3 (38.2) |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

As indicated, the average arsenic concentration and mass rate at the outlet were 22.9 mg/$Nm^3$ (0.01 gr/dscf) and 17.3 kg/hr (38.2 lb/hr), respectively.

3.3.1.2.2 Spray Chamber/Electrostatic precipitator (ASARCO-El Paso). Inlet and outlet arsenic emission measurements were made by EPA across this spray chamber/ESP. Three runs were made at each of three inlet locations and at one outlet location. A summary of the test results is presented in Table 3-12.[37] Measurements were also made for total particulate emissions on two separate occasions. Three inlet locations and one outlet location were sampled. During the first test, two runs were made at each location. During the later test, three runs were made only at the outlet. Tables 3-13 and 3-14 present the test data.[36,37]

Smelting facilities at the ASARCO-El Paso smelter consist of four multi-hearth roasters, a single reverberatory furnace, and three converters. When the tests were conducted, the smelter had the following process emission control configuration. Process gases from the reverberatory furnace passed through two parallel waste heat boilers where they were cooled to a temperature of about 400°C (750°F). The existing gas stream was then combined with the roaster offgases in a large rectangular flue. The combined gas stream, which averaged about 5,100 $Nm^3$/min (180,000 scfm), then entered a spray chamber where it was cooled from about 220°C (428°F) to less than 115°C (240°F). After cooling, the combined roaster and reverberatory gases entered an electrostatic precipitator for particulate removal prior to being discharged into the main stack.* The precipitator consists of seven parallel chambers, each containing four sections, with a total electric field volume of 516 $m^3$ (18,228 $ft^3$).

As indicated in Table 3-12, the average arsenic concentration recorded at the inlet and outlet was 0.308 and 0.006 g/$Nm^3$ (0.13 and 0.0026 gr/dscf), respectively. The average mass rates were 95.9 kg/hr (211 lb/hr) at the inlet and 2.1 kg/hr (4.6 lb/hr) at the outlet. The results indicate that the average arsenic removal efficiency of this unit was 97.8 percent for the three runs conducted.

Table 3-13 presents the particulate matter test results obtained during the first test series conducted on the spray chamber/ESP at ASARCO-El Paso. The average particulate concentration and mass rate recorded at the inlet were 5.11 g/$Nm^3$ (2.23 gr/dscf) and 1,134 kg/hr (2,495 lb/hr), respectively. The average particulate concentration and mass rate at the outlet were, respectively, 0.098 g/$Nm^3$ (0.043 gr/dscf)

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*In current practice, reverberatory furnace gases are treated separately in one ESP while roaster gases are treated in another ESP.

Table 3-12. ARSENIC PERFORMANCE DATA FOR SPRAY CHAMBER/ ELECTROSTATIC PRECIPITATOR AT ASARCO-EL PASO

| Sample run | Arsenic emissions[a] |  |  |  |  |  |  |  | Efficiency, percent |
|---|---|---|---|---|---|---|---|---|---|
|  | Inlet |  |  |  | Outlet |  |  |  |  |
|  | $Nm^3$/min | °C | g/$Nm^3$ | kg/hr | $Nm^3$/min | °C | g/$Nm^3$ | kg/hr |  |
| 1 | 5636 | 194 | 0.100 | 33.8 | 4054 | 102 | 0.005 | 1.5 | 95.6 |
| 2 | 5983 | 191 | 0.338 | 123.1 | 4233 | 104 | 0.009 | 3.4 | 97.2 |
| 3 | 3467 | 252 | 0.594 | 123.8 | 4490 | 105 | 0.004 | 1.5 | 98.8 |
| Avg. | 5028 | 212 | 0.308 | 95.9 | 4259 | 104 | 0.006 | 2.1 | 97.8 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

## Table 3-13. PARTICULATE PERFORMANCE DATA FOR THE SPRAY CHAMBER/ ELECTROSTATIC PRECIPITATOR AT ASARCO-EL PASO

| | Particulate Emissions[a] | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | Inlet | | | | Outlet | | | | |
| Sample run | °C | $Nm^3$/min | g/$Nm^3$ | kg/hr | °C | $Nm^3$/min | g/$Nm^3$ | kg/hr | Efficiency, percent |
| 1 | 226 | 3,741 | 4.54 | 1,022 | 105 | 4,346 | 0.111 | 40.8 | 96.1 |
| 2 | 231 | 3,631 | 4.87 | 1,248 | 105 | 4,700 | 0.085 | 33.8 | 97.3 |
| Avg. | 229 | 3,686 | 5.11 | 1,134 | 105 | 4,523 | 0.098 | 37.2 | 96.7 |

[a] Concentration and mass rate data are based on measurements on the probe, cyclone, and filter catch (front half).

Table 3-14. PARTICULATE PERFORMANCE DATA FOR THE SPRAY CHAMBER/ELECTROSTATIC PRECIPITATOR OUTLET AT ASARCO-EL PASO

| Particulate Emissions[a] | | | | |
|---|---|---|---|---|
| Sample run | $Nm^3$/min | °C | g/$Nm^3$ | kg/hr |
| 1 | 4,346 | 104 | 0.14 | 47.3 |
| 2 | 4,940 | 96 | 0.21 | 73.0 |
| 3 | 4,816 | 104 | 0.09 | 32.3 |
| Avg. | 4,700 | 101 | 0.15 | 50.9 |

[a] Concentration and mass rate data are based on measurements on the probe, cyclone, and filter catch (front half).

and 37.2 kg/hr (81.8 lb/hr). The average particulate removal efficiency measured for the two sample runs performed was 96.7 percent.

Table 3-14 presents particulate matter test results obtained at the outlet of the spray chamber/ESP during the second test series. The average outlet particulate concentration and mass rate values recorded during these latter tests were 0.15 g/Nm$^3$ (0.066 gr/scf) and 50.9 kg/hr (112 lb/hr), respectively; this is about one and one-half times higher than the values recorded during the earlier tests. The higher concentrations and mass rates recorded in the latter tests were likely due to the higher smelter production rates during those tests.

3.3.1.3 Venturi Scrubbers (Kennecott-Hayden). Arsenic emission measurements were conducted by EPA at the Kennecott-Hayden smelter to evaluate the performance of the venturi scrubber used in series with a spray chamber equipped with impingement plates to preclean and condition fluid-bed roaster offgases prior to acid manufacturing. The roaster process gases pass through a series of primary and secondary cyclones where an estimated 95 percent of the entrained calcine is recovered and the gas stream is cooled to about 315°C (600°F). The cyclone exhaust, consisting of about 565 Nm$^3$/min (20,000 scfm) with an estimated dust loading of 57 g/Nm$^3$ (25 gr/scf), then enters the venturi scrubber where most of the particulate matter is collected. Weak acid scrubbing liquor is injected into the venturi throat at a rate of about 1,457 liter/min (385 gpm), resulting in a pressure drop of about 41 cm (16 in.) of water across the throat. Gases exiting the scrubber then enter a spray-type scrubber equipped with perforated plates, where they are humidified and cooled to about 46°C (115°F) prior to being combined with the converter process gases, and subsequently treated in a double-contact acid plant. The pressure drop across both scrubbers is about 61 cm (24 in.) of water.

Both inlet and outlet arsenic emission samples were obtained. The inlet sample was collected upstream of the venturi scrubber while the outlet sample was collected downstream of the spray-type scrubber. It was not possible to sample directly downstream of the venturi scrubber because of the system configuration. Table 3-15 presents a summary of the results.[38]

Table 3-15. ARSENIC PERFORMANCE DATA FOR VENTURI SCRUBBER AT KENNECOTT-HAYDEN

| Sample run | Arsenic emissions[a] | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| | °C | $mg/Nm^3$ | kg/hr | °C | $mg/Nm^3$ | kg/hr | |
| 1 | 336 | 29.53 | 0.85 | 46 | 0.64 | 0.02 | 97.9 |
| 2 | 328 | 25.87 | 0.74 | 44 | 0.27 | 0.01 | 98.9 |
| 3 | 324 | 22.90 | 0.75 | 28 | 0.32 | 0.01 | 98.8 |
| Avg. | 329 | 26.10 | 0.78 | 39 | 0.41 | 0.01 | 98.4 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

As the results indicate, the arsenic inlet loading to the scrubber was quite low, averaging 26.10 $mg/Nm^3$ (0.0114 gr/dscf) and 0.78 kg/hr (1.72 lb/hr). The outlet concentration was very low, averaging 0.41 $mg/Nm^3$ (0.0002 gr/dscf). The arsenic mass emission rate at the outlet averaged 0.01 kg/hr (0.023 lb/hr). The average collection efficiency observed was 98.4 percent. It should be noted that the three inlet runs exceeded isokinetic tolerances (refer to Appendix C). This was due primarily to the large fluctuations in the moisture content of the gas stream from run to run. As a result, the actual inlet concentration was probably somewhat higher than that measured. Consequently, the actual arsenic collection efficiency of the system is probably slightly higher than the 98.4 percent recorded.

3.3.1.4 Sulfuric Acid Plants. Tests were performed at the Kennecott-Hayden, ASARCO-El Paso, and Phelps Dodge-Ajo smelters to evaluate the performance of acid plants in controlling arsenic emissions.

Gas precleaning and conditioning of the smelter offgases used for sulfuric acid manufacturing is absolutely necessary for effective acid plant operation. Both hot and cold gas cleaning devices are used.

3.3.1.4.1 Double-contact acid plant (Kennecott-Hayden). The double-contact acid plant operated at this smelter treats a combination of fluid-bed roaster and converter process gases. Acid production is typically about 935 Mg/day (850 tpd) of 93.5 percent sulfuric acid.

After passing through a series of cyclones for calcine recovery, the fluid-bed roaster offgases are treated in a venturi scrubber for particulate removal and a spray tower for humidification and cooling. The converter offgases are captured in water-cooled hoods, cooled to 370°C (700°F) in a gas cooler, and routed to an electrostatic precipitator for particulate removal. Gases exiting the precipitator enter a spray tower, similar to that used on the roaster gas stream, where they are humidified, cooled, and subsequently combined with the roaster gas stream. The combined gas stream, totaling about 2,120 $Nm^3$/min (75,000 scfm) at 46°C (115°F), then passes through three parallel trains of two mist precipitators in series, where acid mist and any remaining particulates are precipitated. The gas stream, which typically contains 5 to 8 percent $SO_2$, then enters the double-contact acid plant where it is dried, the $SO_2$ converted to $SO_3$, and the $SO_3$ absorbed in weak acid to form strong acid.

Three arsenic test runs were conducted by EPA on the acid plant tail gas stream (outlet). These tests were performed concurrently with those across the venturi scrubber described in the preceding section. The average arsenic concentration measured was 3.43 mg/$Nm^3$ (0.0015 gr/dscf). The corresponding arsenic mass rate averaged 0.41 kg/hr (0.90 lb/hr).[38]

3.3.1.4.2 Double-contact acid plant (ASARCO-El Paso). Offgases generated during converter blowing operations at the ASARCO-El Paso smelter are treated in a 454 Mg/day (500 tpd) double-contact sulfuric acid plant for $SO_2$ removal. The offgases are captured in water-cooled hoods, passed through two parallel waste heat boilers, and cooled by evaporative cooling in a spray chamber. The cooled gases [about 1,700 $Nm^3$/min (60,000 scfm) at 149°C (300°F)] then enter an ESP for particulate removal. The precipitator consists of four parallel chambers, each having four sections in series. The exiting gases pass through a venturi scrubber for additional particulate removal, are humidified and cooled in a pair of packed bed scrubbers, and then are treated in a series of mist precipitators where water and any remaining particulates are removed prior to entering the acid plant.

Arsenic emission measurements were conducted by EPA at the inlet to the spray chamber and at the acid plant outlet. Three sample runs were made on the inlet and four on the outlet. The results are summarized in Table 3-16.[39]

Table 3-16. ARSENIC PERFORMANCE DATA FOR DOUBLE-CONTACT ACID PLANT AT ASARCO-EL PASO

| | Arsenic emissions[a] | | | | | | |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| Sample run | °C | $g/Nm^3$ | kg/hr | °C | $g/Nm^3$ | kg/hr | Efficiency, percent |
| 1 | 222 | 2.36 | 233.3 | 64 | 0.0002 | 0.022 | 99.99 |
| 2 | 209 | 0.228 | 21.4 | 64 | 0.0031 | 0.355 | 98.3 |
| 3 | 200 | 0.262 | 24.4 | 66 | 0.0011 | 0.126 | 99.5 |
| 4 | b | b | b | 69 | 0.0004 | 0.038 | b |
| Avg. | 210 | 0.976 | 96.0 | 67 | 0.0015[c] | 0.168[c] | 99.8 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

[b] Only three inlet sample runs were made.

[c] Average of first three runs only.

As indicated, the measured inlet and outlet arsenic concentrations averaged 0.976 $g/Nm^3$ (0.426 gr/dscf) and 0.0015 $g/Nm^3$ (0.0007 gr/dscf), respectively. The arsenic mass rate averaged 96.0 kg/hr (211 lb/hr) at the inlet and 0.168 kg/hr (0.370 lb/hr) at the outlet, indicating an average arsenic removal efficiency in excess of 99 percent.

3.3.1.4.3 Single-contact acid plant (Phelps Dodge-Ajo). Offgases generated during converting at the Phelps Dodge-Ajo smelter are treated in an ESP for particulate removal followed by a 544 Mg/day (600 tpd) single-contact sulfuric acid plant for $SO_2$ removal. The offgases pass through waste heat boilers where they are cooled to about 315°C (600°F), enter a balloon flue, and then pass through an electrostatic precipitator. The precipitator consists of two independent horizontal parallel units with three fields, each of which is designed to handle 5,490 $m^3$/min (210,000 acfm) at 340°C (650°F) and 95.1 kPa (13.8 psia). The exiting gases pass into the scrubbing section of the acid plant where they are treated in a humidifying tower, a cooling tower, and a mist precipitator. The cleaned gases are then processed in the acid plant. Either 93 or 98 percent sulfuric acid can be produced.

Simultaneous inlet and outlet arsenic emission measurements were conducted by EPA. Three sample runs each were made on the inlet and outlet. The results are summarized in Table 3-17.[40] The offgases treated in the acid plant contained a negligible amount of arsenic. The measured inlet and outlet concentrations averaged 0.00007 g/Nm$^3$ (0.00003 gr/dscf) and 0.000016 g/Nm$^3$ (0.000007 gr/dscf), respectively. The arsenic mass rate averaged 0.004 kg/hr (0.009 lb/hr) at the inlet and 0.001 kg/hr (0.0022 lb/hr) at the outlet, indicating an average arsenic removal efficiency of 75 percent.

Table 3-17. ARSENIC PERFORMANCE DATA FOR SINGLE-CONTACT ACID PLANT AT PHELPS DODGE-AJO

| Sample run | Arsenic emissions[a] | | | | | | Efficiency, percent |
|---|---|---|---|---|---|---|---|
| | Inlet | | | Outlet | | | |
| | °C | g/Nm$^3$ | kg/hr | °C | g/Nm$^3$ | kg/hr | |
| 1 | 190 | 0.00008 | 0.006 | 60 | 0.000007 | 0.0006 | 90.0 |
| 2 | 182 | 0.00003 | 0.002 | 73 | 0.00001 | 0.0007 | 65.0 |
| 3 | 172 | 0.00009 | 0.005 | 53 | 0.00003 | 0.0013 | 64.0 |
| Avg. | 181 | 0.00007 | 0.004 | 62 | 0.000016 | 0.001 | 75.0 |

[a] Concentration and mass rate data are based on measurements on the total catch (front and back half).

### 3.3.2 Fugitive Control Systems Evaluation

#### 3.3.2.1 Local Ventilation Techniques Applied to Calcine Discharge, Matte Tapping, and Slag Tapping.

The performance capability of the local ventilation techniques used at the ASARCO-Tacoma smelter for the control of fugitive arsenic emissions from calcine discharge, matte tapping, and slag tapping operations were evaluated. These techniques were previously described in Sections 3.1.2.4, 3.1.2.5, and

3.1.2.6, respectively. Visual observations were made using either EPA Method 22 or EPA Method 9, depending on whether the emissions observed were intermittent or continuous. Method 22 is used to determine the occurrence of visible emissions, while Method 9 is used to determine the opacity of emissions. A summary of the visible emission data obtained is presented in Table 3-13.[41]

3.3.2.1.1 Calcine transfer. Thirteen calcine transfer operations, each averaging about 2 minutes in duration, were observed. The visual observations were made using EPA Method 22 at the opening of the tunnel-like structure used to house the calcine hoppers and larry cars during the calcine transfer (discharge) operations. As the data indicate, no visible emissions were observed at any time.

3.3.2.1.2 Matte tapping. Visible emission observations during furnace matte tapping were also made using EPA Method 22. Simultaneous but separate observations were made both at the furnace tap port and at the launder-to-ladle transfer point. Sixteen taps, averaging approximately 5.5 minutes in duration, were observed at the tap port. Out of the 16 observations made at the matte tap port, no visible emissions were observed 100 percent of the time on 14 of these, with only slight emissions ranging from 1 to 3 percent of the time for the remaining two observations. No visible emissions were observed 100 percent of the time from the launder-to-matte ladle transfer point during all 15 observations made at the transfer point.

3.3.2.1.3 Slag tapping. Slag tapping emissions were observed using both EPA Methods 22 and 9. As with matte tapping, separate observations were made at the furnace tap port location and at the slag launder-to-slag pot transfer point. Results obtained using EPA Method 22 for eight observations at the slag tap port showed that visible emissions were observed about 5 percent of the time on the average, with the highest single observation showing the presence of visible emissions 15 percent of the time. Visual observations made at the slag launder-to-pot transfer point indicated very poor performance, with visible emissions being observed 72 to 99 percent of the time over 11 slag taps. Additional data obtained using EPA Method 9 showed significant emissions with opacities as high as 50 percent. Conversations

Table 3-18. SUMMARY OF VISIBLE EMISSION OBSERVATION DATA FOR CAPTURE SYSTEMS ON FUGITIVE EMISSION SOURCES AT ASARCO-TACOMA[a]

| Operation | EPA Method 22 | | | | EPA Method 9 | | | |
|---|---|---|---|---|---|---|---|---|
| | No. of readings taken | Average observation time, min:sec | Average percent time emissions observed on all readings | Range of percent time emissions observed | No. of readings taken | Average observation time, min:sec | Average opacity, percent | Range of opacity percent observed |
| Calcine transfer system | 13 | 1:55 | 0 | 0 | | | | |
| Matte tapping: | | | | | | | | |
| at matte tap port and launder | 16 | 5:28 | 0.2 | 0-3 | | | | |
| at matte discharge into ladle | 15 | 5:21 | 0 | 0 | | | | |
| Slag tapping: | | | | | | | | |
| at slag tap port and launder | 8 | 13:38 | 5.3 | 0-15 | 2 | 13:45 | 6 | 0-30 |
| at slag discharge into pots | 11 | 15:27 | 88 | 72-99 | 7 | 14:32 | 12 | 0-50 |

[a] Visible emission observations made on June 24 through 26, 1980.

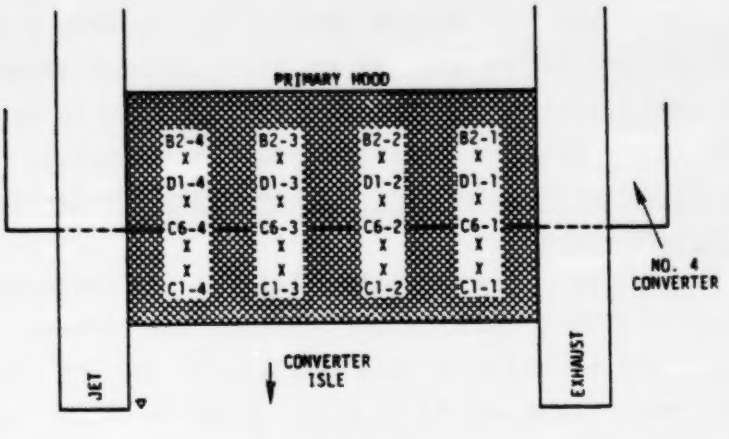
with smelter personnel revealed that the ventilation hood at the slag launder discharge point had been damaged when hit by a truck. Although an inspection of the ventilation hood and ancillary ductwork showed no apparent damage, ventilation at this location was concluded to be inadequate to handle the volume of emissions and fume generated.

#### 3.3.2.2 Fugitive Emission Controls for Converters-Air Curtain Secondary Hood Capture System.

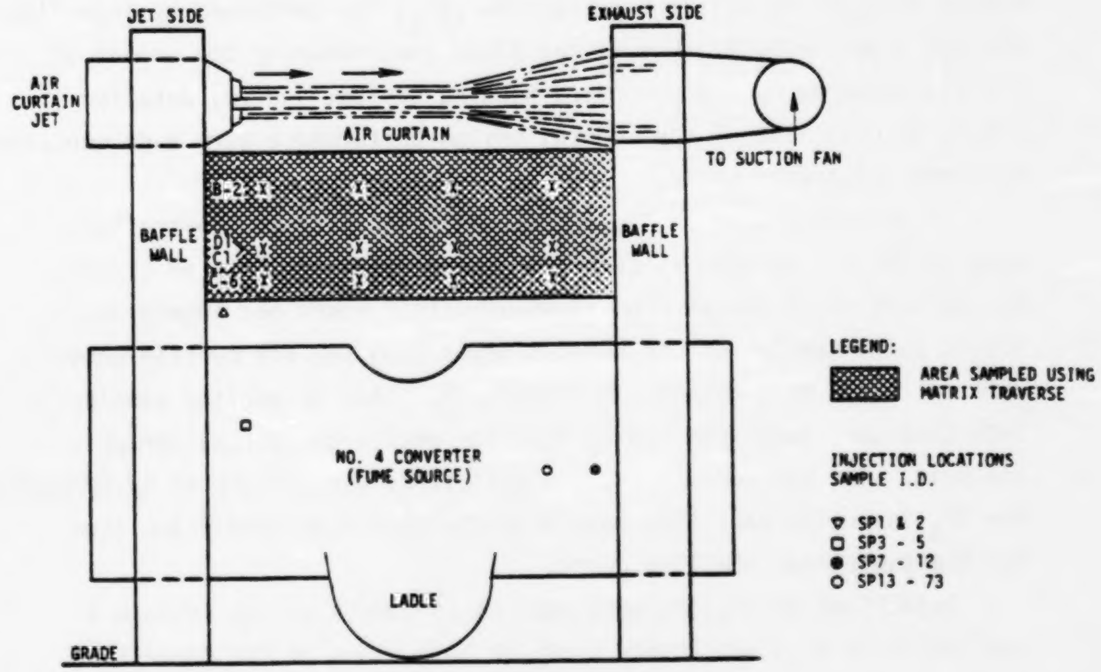
3.3.2.2.1 Evaluation program at ASARCO-Tacoma. EPA conducted an evaluation program in January 1983, on the prototype air curtain secondary hood recently installed at ASARCO-Tacoma.[29] The primary objective of this test program was to obtain an estimate of the overall capture efficiency of the air curtain control system and also the capture efficiency during specific modes of converter operation. Capture efficiencies of the system during three complete converter cycles were estimated using two principal techniques: (1) a gas tracer study using sulfur hexafluoride ($SF_6$) was performed by injecting the gas into the fugitive emission plume and measuring the amount of the gas captured by the secondary hooding system, and (2) detailed visual observations of the hooding system performance were made concurrent with the gas tracer study.

In the gas tracer study, $SF_6$ was injected into the controlled area of the air curtain at constant, known rates of 30 to 50 cc/min for periods which ranged from 15 minutes to 2 hours per injection. Single point samples of the exhaust gases from the air curtain hood were collected at a downstream sampling location by pulling samples into 15-liter, leak-free Tedlar bags for onsite gas chromatographic analysis. The air curtain capture efficiency was calculated by comparing the $SF_6$ injection mass flow rate with the mass flow rate calculated for the downstream sampling point.

Injections of $SF_6$ gas were made at 16 sample points through 4 test ports in adjacent access doors on both sides of the converter baffle walls. The locations of the points are shown in Figures 3-18 and 3-19. In addition to the efficiency measurements made for the points in the primary testing area, several tests were performed at injection points outside of this area (below the converter centerline) in


PRIMARY HOOD
B2-4
D1-4
C6-4
C1-4
B2-3
D1-3
C6-3
C1-3
B2-2
D1-2
C6-2
C1-2
B2-1
D1-1
C6-1
C1-1
NO. 4 CONVERTER
CONVERTER ISLE
JET
EXHAUST

TOP VIEW


JET SIDE
EXHAUST SIDE
AIR CURTAIN JET
AIR CURTAIN
TO SUCTION FAN
B-2
D1
C1
C-6
BAFFLE WALL
BAFFLE WALL
NO. 4 CONVERTER (FUME SOURCE)
LADLE
GRADE
LEGEND:
AREA SAMPLED USING MATRIX TRAVERSE
INJECTION LOCATIONS SAMPLE I.D.
▽ SP1 & 2
□ SP3 - 5
● SP7 - 12
○ SP13 - 73

ELEVATION

Figure 3-18. $SF_6$ Tracer Injection Locations

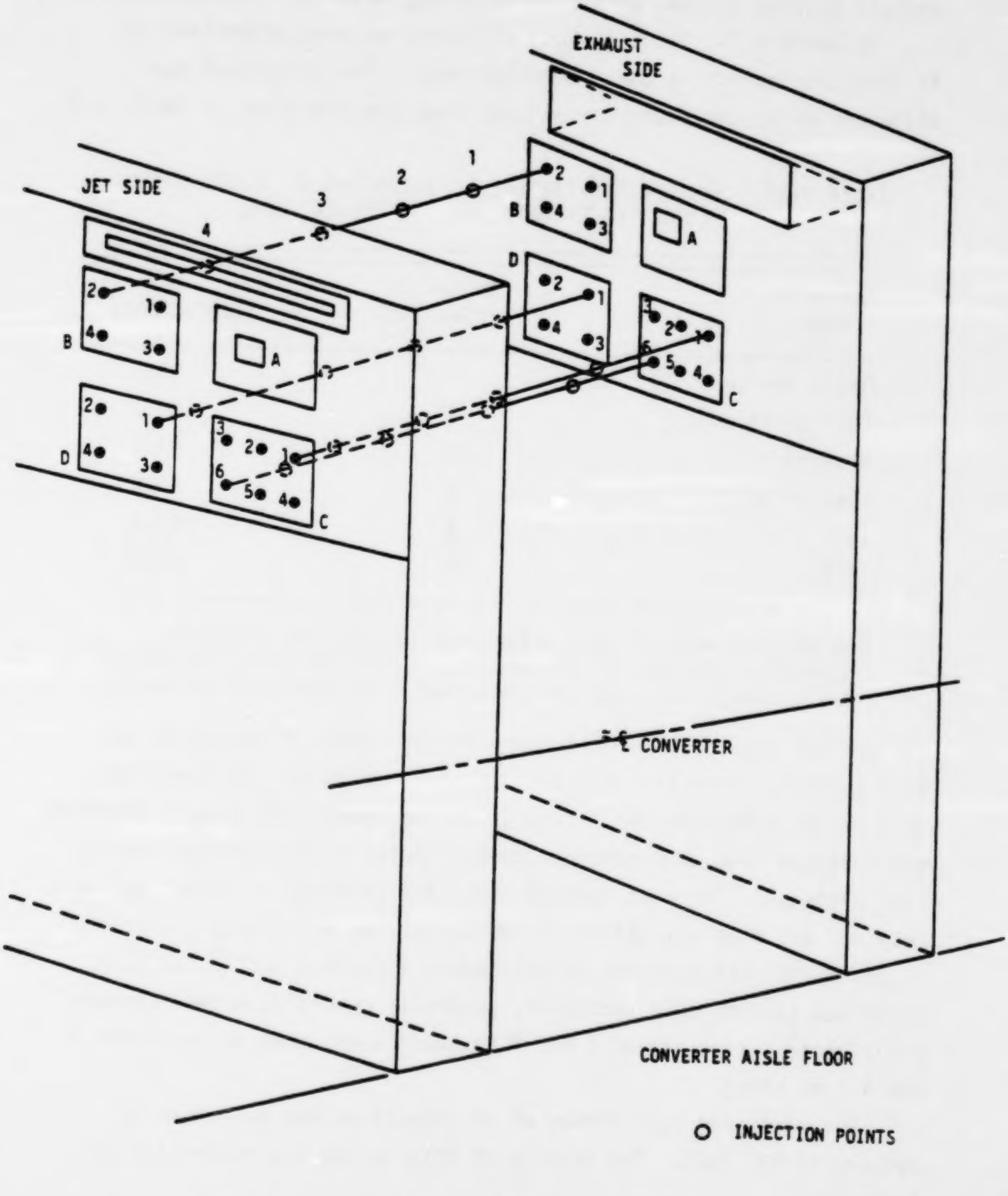

EXHAUST SIDE
JET SIDE
1
2
3
4
A
B
C
D
1
2
3
4
5
6
℄ CONVERTER
CONVERTER AISLE FLOOR
○ INJECTION POINTS

Figure 3-19. Tracer Injection Test Ports

an attempt to characterize the effective capture area of the air curtain hooding system, particularly during converter rollout activities.

On January 14, 1983, capture efficiencies were determined for 45 injection points in the controlled area. The calculated mean efficiencies by converter operational mode are presented in Table 3-19.

Table 3-19. AIR CURTAIN CAPTURE EFFICIENCIES AT ASARCO-TACOMA USING GAS TRACER METHOD - JANUARY 14, 1983

| Converter mode | Number of injections | Mean efficiency |
|---|---|---|
| Matte charge | 7 | 93.1 |
| Cold addition | 3 | 102.0 |
| Blowing | 19 | 92.8 |
| Slag skimming | 9 | 95.0 |
| Idle | 7 | 93.4 |
| TOTAL | 45 | 93.5[a] |

[a] Calculated overall mean efficiency assumes the converter operation consists of 80 percent blowing and idle, 15 percent matte charge and cold addition, and 5 percent slag skimming.

The overall mean capture efficiency for all modes of operation was 93.5 percent. With the exception of cold additions, the operating mode of the converter had little effect on capture efficiency measured, which ranged from 92.8 percent during blowing to 95.0 percent during slag skimming. The port through which the releases of tracer gas were made did not have any effect on the calculated efficiency. However, it was found that sampling points tested through a particular port exhibited considerable variation, generally recording higher capture efficiencies at positions 1 and 2 (exhaust side) than at positions 3 and 4 (jet side).

The remaining test series of 48 injections was performed on January 17-18, 1983. The results of this series are summarized in Table 3-20.

### Table 3-20. AIR CURTAIN CAPTURE EFFICIENCIES AT ASARCO-TACOMA USING GAS TRACER METHOD - JANUARY 17-19, 1983

| Converter mode | Number of injections | Mean efficiency |
|---|---|---|
| Matte charge | 6 | 94.2 |
| Cold addition | 3 | 96.7 |
| Blowing | 27 | 96.7 |
| Slag skimming | 7 | 94.3 |
| Copper pour | 4 | 88.5 |
| Idle | 4 | 100.0 |
| TOTAL | 51 | 96.5[a] |

[a] Calculated overall mean efficiency assumes the converter operation consists of 80 percent blowing and idle, 15 percent matte charge and cold addition, and 5 percent slag skimming and copper pour.

The overall mean capture efficiency for all operational modes was 96.5 percent. As with the data recorded on January 14, the operating mode appeared to have no significant effect on the individual calculated efficiencies, which ranged from 88.5 percent during copper pouring to 96.7 percent during both blowing and cold additions. However, any consistent, small variations in the efficiencies for various modes, if they were present, would be difficult to detect in the relatively small number of test runs (injections) which were made. The error in the calculated air curtain capture efficiencies has been estimated to be ±18 percent.[42] For this second test series, it was also found that the location of the test port had no effect on efficiency, while exhaust-side efficiencies were found to be somewhat higher than jet-side efficiencies. Test results from the injection points in these tests indicate that, on the average, about 95 percent of the gases and particulate matter in the area immediately above the converter is likely to be captured by the air curtain secondary hooding system.

In addition to these two test series, a series of special injection point tests was conducted in order to assess the effective capture area of the secondary hood system outside the confines of the hood. The special injection tests were performed with the injection probe at a number of points on the perimeter of the main test area, such as very close to the baffle wall and below the ladle during the matte charging and cold addition modes. Table 3-21 shows the results of this test series.

Table 3-21. AIR CURTAIN CAPTURE EFFICIENCIES AT ASARCO-TACOMA FOR SPECIAL GAS TRACER INJECTION POINTS - JANUARY 18-20, 1983

| Converter mode | Number of injections | Mean efficiency |
|---|---|---|
| Matte charge | 17 | 61.8 |
| Cold addition | 6 | 61.5 |
| Blowing | 6 | 33.0 |
| Slag skimming | 28 | 84.0 |
| Copper pour | 4 | 80.8 |
| Idle | 8 | 53.8 |
| TOTAL | 69 | 49.4[a] |

[a] Calculated overall mean efficiency assumes the converter operation consists of 80 percent blowing and idle, 15 percent matte charge and cold addition, and 5 percent slag skimming and copper pour.

The overall average capture efficiency for the 69 special injection points was 49.4 percent. Unlike the first two test series, the capture efficiency in the special series was sensitive to converter mode. For example, the slag skimming and copper pour efficiencies are higher, at 84.0 and 80.8 percent, respectively, than the other modes because of the position of the ladle (above the injection probe) during these modes.

During the course of the gas tracer study, from January 18 to 22, 1983, detailed visual observations were made of the performance of the air curtain control system through the various converter operating modes.[43] The purpose was to estimate capture effectiveness and to qualitatively characterize both captured emissions and emissions escaping capture.

An important benefit from these observations which could not be derived from the results of the gas tracer study was the conclusion that operating practices play a major role in the overall performance of the control system. The crane operator who moves the ladle into position for charging, skimming, or pouring has some latitude in the positioning of the ladle, the speed of pouring, and the speed and timing of the movement of the ladle to and from the converter. Observations show that slower, more deliberate movements and closer positioning of poured materials to the converter have a definite effect on the extent of air curtain penetration and spillage into the converter aisle.

The practices employed by crane and converter operators were found to vary significantly throughout the observation period. For example, during slag skimming, it was observed that the rate of pouring varied for individual skims. The capture efficiency was higher when the pour rate was lower, since a faster rate would sometimes "overwhelm" the control system. Also, the position of the ladle was noticed to be important, because when the ladle was positioned close to the converter mouth the capture efficiency was enhanced. During matte charging, the withdrawal of the crane from above the converter was often observed to cause "drag-out" emissions, especially when the crane was moved immediately after the converter was charged. When the crane was left in place for a few moments after charging (and the heaviest part of the emissions had risen to the air curtain), the drag-out emissions were noticeably reduced, and the capture effectiveness was improved.

Only one period of blowing was evaluated quantitatively during the observations. Some penetration of the air curtain was noticed (5 to 10 percent) during roll-in when the blowing air first started, but the overall capture efficiency was judged to be about 95 percent.

Overall capture efficiencies for individual matte chargings were in the range of 90 to 95+ percent. Cold additions (adding of cool, solidified materials) to the converter frequently produced emissions heavy enough to virtually overwhelm the capture system, especially when a fire ignited in the converter. Capture efficiencies were somewhat lower overall than for matte charging, typically ranging from 80 to 95 percent.

During slag skimming, it was observed that the rate of pouring varied for individual skims. For pour rates judged to be slow or moderate, efficiencies generally ranged from 80 to 95 percent. For faster pour rates, the typical efficiencies dropped to 70 to 80 percent.

Copper pouring generally produced a moderate to heavy amount of fume; both air curtain penetration and spillage out into the converter aisle were very slight. Capture efficiencies were typically 90 to 95 percent during pouring. However, at initial roll-out prior to pouring, the efficiency could be as low as 70 percent.

The observation that fumes would frequently spill out into the converter aisle (especially during slag skimming activities) indicates that efficiencies calculated by the gas tracer method may be too high in some instances. The gas tracer method is based on the assumption that the entire fugitive emission plume either is captured by the air curtain exhaust or penetrates the curtain vertically (since vertically escaping gases should contain a homogeneous concentration of the tracer gas). Therefore, by not accounting for emissions which escape before approaching the air curtain, the gas tracer method probably overpredicted the capture efficiency achieved during slag skimming operations.

In general, visual assessments of secondary hood capture effectiveness correlate quite well with the average efficiencies determined by the gas tracer method. As mentioned earlier, operating practices have a significant influence on the degree of capture achieved during any individual converter operation, and many of the extreme observed values can be understood in terms of the operating practices employed during those particular operations.

3.3.2.2.2 Visible Emissions Observations at Tamano Smelter. All three converters at the Tamano smelter in Japan are equipped with a fixed enclosure and air curtain system for control of fugitive emissions generated during various modes of converter operation. The enclosure doors and roof are kept open and the air curtain system is turned on during the matte charging. During all other modes of the converter operation the doors and roof are kept closed and the air curtain system is turned off. This system is described in Section 3.1.2.7.2.

Visible emission observations were made for the air curtain secondary hood operated on the No. 3 converter during day-shifts on March 12 and March 13, 1980. The converter is a conventional Pierce-Smith design, measuring about 9 meters in length and 4 meters in diameter. Observations were made using EPA Methods 22 and 9, depending on whether the emissions observed were intermittent or continuous, for the different modes of converter operation comprising a converter cycle. A discussion of the results obtained during each mode of converter operation follows.

Matte charging. Usually three ladles of matte are brought to the converter and charged in a 10- to 30-minute period. Actual matte charging from each ladle lasts for 1 to 1.5 minutes. The fixed enclosure doors and roof are opened; the air curtain system is turned on; and the ladle of matte is brought into the secondary hood by an overhead crane. The converter is rolled down to an inclined position; the matte ladle is lifted up by the crane; and matte is charged into the converter. At the completion of matte charge, the ladle is moved out of the enclosure and, if needed, another ladle of matte is brought in. After the matte additions are completed, the converter is rotated into the primary converter hood, the roof and doors are closed, and slag blowing is commenced.

Three separate matte charges were observed using both EPA Methods 9 and 22 simultaneously, and one matte charge was observed using EPA Method 9 only. Visual observations for each matte charge observed were made only during the period when the matte was actually flowing into the converter. Results of the visual observations obtained are summarized in Table 3-22.[44]

As shown in Table 3-22, visible emissions were observed for three individual matte charges. The observations ranged from 44 to 77 percent of the time (EPA Method 22). Although somewhat continuous, the opacity results indicate that these emissions were generally slight, typically ranging from 0 to 10 percent opacity, with the highest average opacity recorded for a single matte charge being 25 percent. When present, the emissions appeared as small puffs which penetrated the air curtain stream.

Table 3-22. VISIBLE EMISSIONS OBSERVATION DATA
FOR CONVERTER SECONDARY HOOD SYSTEM
DURING MATTE CHARGING AT THE TAMANO SMELTER

| Sample run | Method 22 | | Method 9 | | |
|---|---|---|---|---|---|
| | Observation period, min | Percent of time emissions observed | Observation period, min | Average opacity for observation period, percent | Range of individual readings |
| 1 | 1.5 | 44 | 1.5 | 5.0 | 0 to 25 |
| 2 | 1.25 | 56 | 1.25 | 4.0 | 0 to 10 |
| 3 | 1.75 | 77 | 1.75 | 3.0 | 0 to 10 |
| 4 | - | - | 1.5 | 0 | - |
| Total | 4.50 | 60 | 6.25 | 2.8 | 0 to 25 |

Slag blow and copper blow. During slag blowing and copper blowing, the converter mouth is enclosed by the primary duct, and offgases are directed to the acid plant. The converter secondary hood doors and roof are closed, and the air curtain system is turned off. Fugitive emissions generated during blowing as a result of primary hood leaks are captured inside the converter housing and are vented to a baghouse for collection. The slag blow, which is divided into three segments, lasts for about 150 minutes per converter cycle and the copper blow for about 200 minutes per cycle.

Visible emission observations were made using EPA Method 9 for the converter hood system for 30 minutes during the slag blow and for 27 minutes during the copper blow. No visible emissions (zero percent opacity) were observed at any time.[44]

Slag discharge. At the end of each of the three slag blow phases, slag is skimmed into a ladle and transported to a sand bed area for cooling. Because of the quantities involved, slag is discharged from the converter two times after the first slag blow and once after the second and third. Each slag skim lasts for about 10 minutes. During each skim, an empty ladle is brought into the enclosure by an overhead crane and placed on the ground in front of the converter. The crane is moved out, and the enclosure doors and roof are closed. The converter is rolled down, and slag is poured into the ladle. After the slag skimming is completed, the converter is rotated upward slightly, the enclosure doors are opened, and the slag ladle is moved out.

Only two skims were observed. The first, which lasted 11 minutes, was observed using EPA Methods 22 and 9. The second slag skim, lasting 9 minutes, was observed using EPA Method 22 only. Each observation period began as the converter started rolling down to pour the slag into the ladle and lasted until the pouring was completed and the converter started rolling up. During the first slag skim observed, no visible emissions were observed at any time. In contrast, during the second slag skim, visible emissions were observed 100 percent of the time. Most of the time, however, these emissions were slight, ranging from 5 and 10 percent opacity and consisting of small puffs which escaped from the enclosure through a narrow opening between the front doors and the enclosure roof.[44]

Blister discharge. At the end of copper blowing, the product blister is discharged into a ladle and transported to a refining furnace. Usually four ladles of blister are filled per converter cycle. Each of the first three blister pours lasts about 12 to 14 minutes with the final pour lasting about 4 minutes. The time elapsed between each blister pour is about 8 to 15 minutes.

At the end of a copper blow, the secondary hood doors and roof are opened. An empty ladle is brought into the secondary hood by the overhead crane and placed in front of the converter. The crane is moved out, and the secondary hood doors and roof are closed. The converter is rolled down, and blister is poured into the ladle. After the blister pour is completed, the converter is rolled up slightly; the hood doors are opened; the blister ladle is taken to the refining furnace by the crane; and the hood doors and roof are closed.

Four blister discharges were observed. Both EPA Methods 22 and 9 were used. A summary of the results is presented in Table 3-23.[44] Although the observation periods used in obtaining the EPA Method 22 data were variable (i.e., different start and end times), the results nonetheless indicate that visible emissions during blister discharge were generally continuous. The EPA Method 9 data, which were obtained only during periods when the blister copper was actually being poured, show that the visible emissions observed were somewhat more substantial than those observed during either matte charging or slag skimming. As shown in Table 3-23, the highest average opacity recorded for a single blister pour was 13 percent, with individual opacity readings ranging from 0 to 35 percent. Again, as with slag skimming, the emissions observed generally appeared above the narrow opening between the front doors of the enclosure and the enclosure roof.

Table 3-23. VISIBLE EMISSIONS OBSERVATION DATA FOR BLISTER DISCHARGE AT THE TAMANO SMELTER

| Sample Run | Method 22 | | Method 9 | | |
|---|---|---|---|---|---|
| | Observation period, min. | Percent of time emissions observed | Observation period, min | Avg. opacity for observation period, percent | Range of individual opacity readings |
| 1 | 25[a] | 42 | - | - | - |
| 2 | - | - | 15.0 | 6.2 | 0 to 30 |
| 3 | 15[b] | 86 | 12.0 | 13.0 | 0 to 35 |
| 4 | 6[c] | 19 | 3.5 | 3.2 | 0 to 25 |
| Total | 15.3 | 49 | 30.5 | 8.5 | 0 to 35 |

[a] Observations started when secondary hood doors opened 12 minutes prior to the blister discharge, during which time the converter body was hit by a vibrating ram.

[b] Observations started with the blister discharge and continued for 3 minutes after completion of the blister discharge.

[c] Observations started with the blister discharge and continued for 2-1/2 minutes after completion of the blister discharge.

3.3.2.3 Building Evacuation and Baghouse (ASARCO-El Paso) - Fugitive emissions from converters and anode furnaces at the ASARCO-El Paso smelter are captured by evacuating the converter building. The building evacuation system at this smelter is described in Section 3.1.2.7.3. The captured fugitive gases are drawn through four openings at the roof of the converter building into ducts which merge into a main duct leading to a baghouse, then through fans to the 250 m (828 ft) main stack.

The fugitive gas flow through the baghouse averages about 14,100 $Nm^3/min$ (498,000 scfm). The baghouse consists of 12 compartments. Normally all compartments are in operation except that one compartment is taken off during the cleaning cycle and another compartment during the maintenance cycle. Each compartment contains 384 Orlon or Dacron bags, 20 cm (8 in.) in diameter and 6.7 m (22 ft) long, providing a cloth area of 1,644 $m^2$ (17,700 $ft^2$) per compartment. The total net cloth area of the baghouse is about 19,732 $m^2$ (212,400 $ft^2$). The baghouse was designed to effectively treat 15,280 $m^3/min$ (540,000 acfm) at 54°C (130°F) with an air-to-cloth ratio of 0.91 $m^3/min$ per $m^2$ (3 $cfm/ft^2$). Mechanical shakers are used for cleaning the bags.

Inlet and outlet emission measurements for inorganic arsenic and total particulate were conducted by EPA across the baghouse. During all tests, converter operations were monitored and testing was conducted only when one or more converters were in operation. The arsenic results obtained are summarized in Table 3-24.[45] As indicated, the measured inlet and outlet arsenic concentrations averaged 3.27 $mg/Nm^3$ (0.0014 gr/scf) and 0.137 $mg/Nm^3$ (0.00006 gr/scf), respectively. The arsenic mass rate averaged 2.92 kg/hr (6.45 lb/hr) at the inlet and 0.111 kg/hr (0.244 lb/hr) at the outlet, indicating an average arsenic removal efficiency in excess of 96 percent. The results of the particulate measurements obtained are summarized in Table 3-25.[46] As shown in the table, the mass particulate inlet concentration was low, averaging only 0.062 $g/Nm^3$ (0.027 gr/scf). Nonetheless, the mass particulate emission rate was relatively high, averaging over 50 kg/hr (110 lb/hr). The low inlet concentration is a result of the large quantities of dilution air associated with the application of general ventilation techniques. The outlet concentration and mass rate averaged 5.1 $mg/Nm^3$ and 3.9 kg/hr, respectively. Although the collection efficiency obtained over three test runs averaged only about 90 percent, the results indicate that collection efficiencies as high as 99 percent are achievable.

### 3.3.3 Conclusions

3.3.3.1 Process Controls. As discussed in Section 3.1.1.1, the arsenic control devices considered as best available control for process sources at primary copper smelters incorporate gas stream

Table 3-24. ARSENIC DATA FOR CONVERTER BUILDING BAGHOUSE AT ASARCO-EL PASO

| Sample run | Arsenic emissions |  |  |  |  |  | Efficiency, percent |
|---|---|---|---|---|---|---|---|
|  | Inlet |  |  | Outlet |  |  |  |
|  | °C | mg/$Nm^3$ | kg/hr | °C | mg/$Nm^3$ | kg/hr |  |
| 1 | 38 | 6.21 | 5.51 | 38 | 0.39 | 0.305 | 94.5 |
| 2 | 36 | 2.09 | 1.96 | 37 | 0.017 | 0.017 | 99.1 |
| 3 | 51 | 1.53 | 1.31 | 51 | 0.015 | 0.012 | 99.1 |
| Avg. | 42 | 3.27 | 2.92 | 42 | 0.137 | 0.111 | 96.2 |

Table 3-25. PARTICULATE DATA FOR CONVERTER BUILDING BAGHOUSE AT ASARCO-EL PASO

| Sample run | Particulate emissions |  |  |  |  |  | Efficiency, percent |
|---|---|---|---|---|---|---|---|
|  | Inlet |  |  | Outlet |  |  |  |
|  | °C | mg/$Nm^3$ | kg/hr | °C | mg/$Nm^3$ | kg/hr |  |
| 1 | 16 | 60.3 | 44.7 | 20 | 11.6 | 10.4 | 76.7 |
| 2 | 22 | 53.3 | 46.3 | 14 | 2.5 | 0.92 | 98.0 |
| 3 | 16 | 70.5 | 61.2 | 16 | 1.1 | 6.4 | 99.3 |
| Avg. | 18 | 61.3 | 50.7 | 17 | 5.1 | 3.9 | 91.3 |

precooling as an integral part of the overall control system. Cooling is vital to effective control because the relatively low saturation concentration of arsenic trioxide at lower temperatures allows a greater percentage of the arsenic to condense into a form which can be collected in a particulate control device. The process control systems tested and discussed in Section 3.3.1 include the spray chamber/baghouse system at Anaconda; the roaster and arsenic plant baghouses at ASARCO-Tacoma; the spray chamber/ESP at ASARCO-El Paso; the reverberatory furnace ESP at ASARCO-Tacoma; the venturi scrubber at Kennecott-Hayden; and the acid plants at ASARCO-El Paso, Kennecott-Hayden, and Phelps Dodge-Ajo. The inlet temperature to the baghouses and the electrostatic precipitators ranged between 80 and 115°C (180 to 240°F).

The results of these tests, in terms of the arsenic collection efficiences recorded, are shown in Figure 3-20. (The results of ESP testing at ASARCO-Tacoma are not presented in Figure 3-20, because the data were collected only for the ESP outlet; and the results of the single-contact acid plant at Phelps Dodge-Ajo, are not presented because of its measured 75 percent efficiency due to extremely low inlet arsenic loadings.) Each black circle in the figure represents a sample run performed on the control device designated at the bottom of the figure. The average efficiencies (designated by horizontal bars) ranged from a low of about 96 percent (baghouse at 84°C) to a high of about 99.7 percent (baghouse at 91°C). The data demonstrate that baghouses, ESP's, venturi scrubbers, and acid plants can be used to provide a high level of control for arsenic emissions. However, as discussed earlier, the collection efficiency for arsenic of any particulate control device can vary depending on the distribution between the particulate and vapor form of the arsenic which reaches the device. This distribution in turn depends on the arsenic concentration in the gas stream and the stream temperature. Therefore, any discussion of control efficiency of a particular type of device must consider these parameters before an estimate of the expected efficiency can be made. The process gas streams entering the control devices tested clearly had sufficiently high concentrations of arsenic trioxide for high control efficiencies to be achieved.

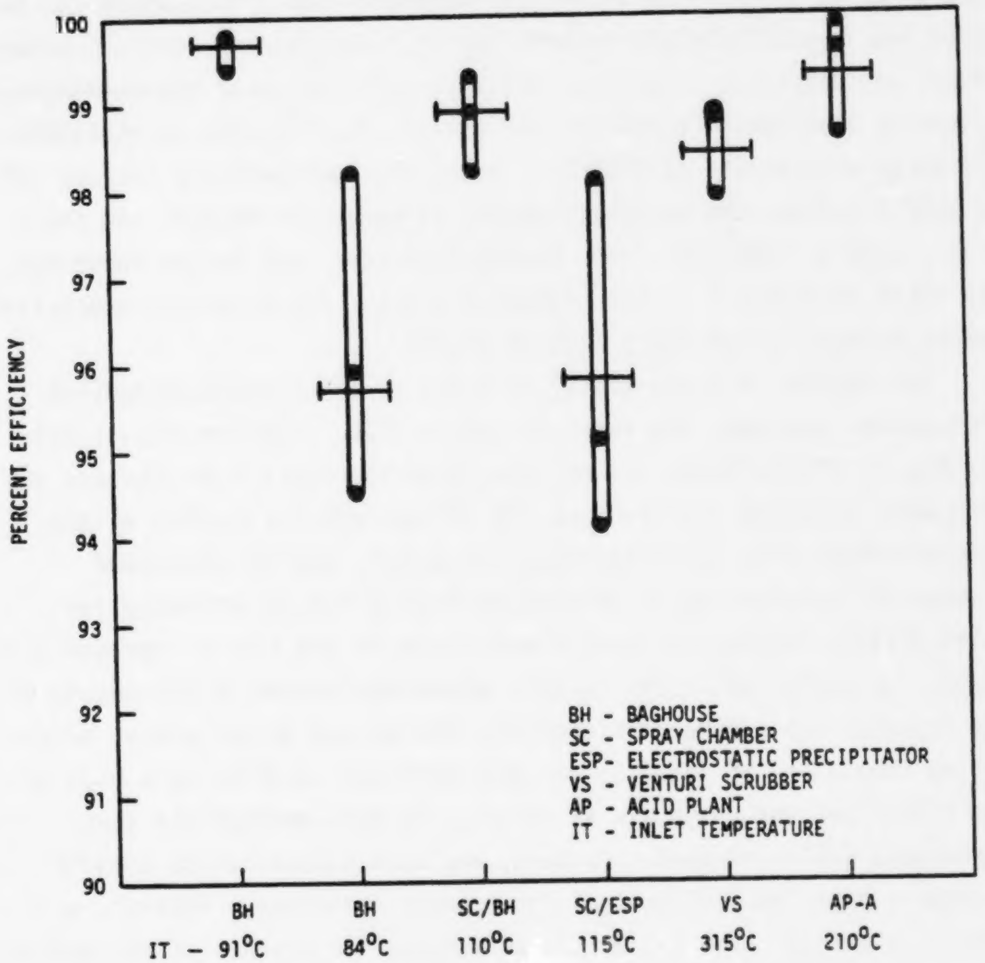


Figure 3-20. Control Device Arsenic Collection Efficiencies

3.3.3.2 Fugitive Controls. The performance capabilities of capture systems for the control of fugitive arsenic emissions from calcine discharge, matte tapping, slag tapping, and converter operations were evaluated. Estimates of the capture efficiency of these systems are based on the visible emissions observations reported in the preceding sections and on subjective judgment. Also evaluated was the performance capability of a baghouse control device used to collect captured fugitive emissions.

Visual observations made on the local ventilation system applied to calcine discharge operations at ASARCO-Tacoma resulted in no visible emissions being observed at any time during the observation period. As a result, it is concluded that such a system is readily capable of achieving a capture efficiency of 90 percent or better.

Observations conducted on the local ventilation system applied to matte tapping operations at the same smelter showed no visible emissions occurring at any time at the matte launder-to-ladle transfer point and only slight emissions of short duration, a maximum of 3 percent of the time, for any individual matte tap observed at the matte tap port. It is concluded, based on these data, that a properly designed and operated ventilation system applied to matte tapping operations should achieve a minimum capture efficiency of 90 percent.

Similar observations made on the ventilation system serving the slag tapping operations at Tacoma showed substantially poorer performance, especially at the slag launder-to-slag pot transfer point where visible emissions were observed nearly 100 percent of the time during each individual slag tap. Based on the results of the visual observations and the fact that the capture system had reportedly been damaged, it is concluded that the slag tapping ventilation system observed at Tacoma, as it was operating at the time, should not be considered representative of a best system of emission reduction. Although slag tapping operations do represent a somewhat more difficult control situation than matte tapping, the outstanding performance demonstrated by the matte tapping controls at Tacoma strongly suggest that a properly designed and operated ventilation system applied to slag tapping operations should be capable of achieving at least 90 percent capture.

Capture systems evaluated for the control of fugitive arsenic emissions from converter operations included the prototype air curtain secondary hoods applied at the ASARCO-Tacoma smelter and the Mitsui Smelting Company smelter in Tamano, Japan, and the general ventilation or building evacuation system applied at the ASARCO-El Paso copper smelter.

The gas tracer study performed on the converter air curtain system at ASARCO-Tacoma indicates that approximately 95 percent of the converter fugitive emissions from all operating modes is captured by the system. Visual observations of the operation of this system verify this measured capture efficiency, but also indicate that operating practices play an important role in attaining this efficiency consistently. EPA has concluded that this efficiency can be achieved with a properly operated air curtain secondary hooding system in combination with proper operating practices.

The visual observations made at Tamano on the air curtain secondary hood indicate that the application of such a system on a conventional Pierce-Smith converter should result in an overall capture efficiency of at least 90 percent. As reported previously, no visible emissions were observed during either slag blowing or copper blowing, which in combination account for nearly 80 percent of a typical converter cycle. Although emissions were observed penetrating the air curtain during matte charging, these emissions were judged to be negligible, consisting of small puffs ranging from 0 to 10 percent opacity. Emissions observed during (slag and blister) pouring operations were somewhat more substantial, varying from 0 to 35 percent opacity. However, as previously indicated, these emissions were generally observed to escape through a narrow opening which existed between the enclosure doors and roof. The smelter representatives indicated that a tighter seal between the doors and roof would result in a significant improvement.

Conclusions regarding the potential effectiveness of the building evacuation system used at the ASARCO-El Paso smelter are based primarily on engineering judgment. Providing the building is properly enclosed and adequate ventilation rates are applied, essentially 100 percent capture should be possible. However, due to the need for openings in the building for access and makeup air, a more conservative estimate of 95 percent capture is considered more reasonable. The fact that this system has shown less than satisfactory performance at the El Paso facility highlights the difficulty of designing an evacuation system which will provide a controlled ventilation air supply for all of the emission sources in the building. Fugitive sources have generally been found to be more successfully controlled by the use of local ventilation (hooding) techniques.

With regard to the collection of fugitive arsenic emissions in a control device, the emission measurements conducted across the baghouse facility serving the converter building evacuation system at ASARCO-El Paso showed that an arsenic collection efficiency of 96 percent or higher was readily achievable, even though the measured inlet concentrations were extremely low, averaging only 3.3 mg/Nm$^3$ (0.0014 gr/scf).

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41. Reference 15, p. 4.

42. Reference 29, p. 4-66.

43. Vervaert, A. and J. Nolan, U.S. Environmental Protection Agency and Puget Sound Air Pollution Control Agency. Log Book Nos. 1 and 2, Observations of Converter Secondary Hood Test at ASARCO, Tacoma. January 18-22, 1983.

44. Reference 22, p. 3.

45. Reference 37, p. 7-8.

46. Reference 37, p. 4-5.

# 4.0 MODEL PLANTS, REGULATORY BASELINE, AND REGULATORY ALTERNATIVES

Arsenic has been listed as a hazardous air pollutant under Section 112 of the Clean Air Act (National Emission Standards for Hazardous Air Pollutants). To protect public health from unreasonable risks associated with exposure to airborne arsenic, standards are being developed to decrease arsenic emissions from primary copper smelters which process high arsenic feed. This section defines the model plants, presents the alternative ways that EPA can regulate arsenic emissions from the affected sources at these primary copper smelters, and defines the regulatory baseline for analysis of the environmental, economic, and energy impacts of the regulatory alternatives on the industry.

## 4.1 MODEL PLANTS

As discussed in Section 1.1, only the ASARCO-Tacoma smelter has been categorized as a smelter which processes materials containing high concentrations of arsenic. Annually, this smelter processes a greater quantity of arsenic in its feed material than the combined total at the other 14 smelters. Since the ASARCO-Tacoma smelter is the only smelter under consideration in this study, a single model plant representative of the ASARCO-Tacoma smelter was developed. Site-specific parameters associated with the copper smelting configuration for this smelter are presented in Table 4-1.

## 4.2 BASELINE

This document evaluates the technical, environmental, energy, cost, and economic impacts of regulatory alternatives for the control of arsenic from high arsenic throughput primary copper smelters. These are measured as incremental changes beyond the situation which would exist in the absence of NESHAP regulations for arsenic. These conditions

Table 4-1. MODEL PLANT PARAMETERS

| Parameter | ASARCO-Tacoma |
|---|---|
| 1. Production capacity (blister copper), Mg/yr (tons/yr) | 91,000 (100,000) |
| 2. Raw material: | |
| Materials used | copper concentrates, lead smelter by-products, smelter reverts, and others |
| Maximum total arsenic content of feed (concentrates, ores, speiss, flux), percent wt | 4.0 |
| kg/hr (lb/hr) | 991 (2,185) |
| 3. By-products | Arsenic trioxide, metallic arsenic, sulfuric acid, liquid $SO_2$ |
| 4. Process equipment: | |
| ● Roasters: No. and type | 6 Herreshoff and 4 C & W multi-hearth roasters[a] |
| Dimensions | 5.79 m(19 ft) diameter by 7.61 m (25 ft) high, 294 $m^2$ (3,160 $ft^2$) hearth area |
| Feed capacity per roaster, Mg/day (tons/day) | 1,087 (1,200) |
| ● Smelting furnaces: | |
| No. and type | 2 reverberatory furnaces |
| Dimensions | No. 1 furnace: 9.14 m by 33.53 m (30 ft by 110 ft)<br>No. 2 furnace: 9.75 m by 35.53 m (32 ft by 110 ft) |
| Feed capacity per furnace, Mg/day (tons/day) | 1,087 (1,200) |

Table 4-1. MODEL PLANT PARAMETERS (Concluded)

| Parameter | ASARCO-Tacoma |
|---|---|
| ● Converters: | |
| No. and type | 4 Pierce-Smith converters[b] |
| Dimensions | 3 converters:<br>3.9 m diameter by 9.14 m length (13 ft diameter by 30 ft length)<br>1 converter:[c]<br>3.35 m diameter by 7.93 m length (11 ft diameter by 26 ft length) |
| ● Anode Furnaces: | |
| No. and type | 2 Tilting Anode furnaces, 150 tons/charge[d] |
| ● Arsenic Plant: | |
| Process Equipment | 3 Godfrey Roasters<br>3 Arsenic Trioxide settling kitchens<br>2 arsenic metallic furnaces and condensers |

[a] Normally six or seven used at a time.

[b] Normally one converter will be on copper blow. One converter is on standby or is used as a holding furnace.

[c] To be enlarged.

[d] Normally one anode furnace is used as a holding furnace.

are commonly referred to collectively as a baseline. This section defines such a baseline as it applies to the primary copper smelting industry in general and to the ASARCO-Tacoma smelter specifically.

### 4.2.1 Regulatory Considerations

The ASARCO-Tacoma smelter is subject to existing Federal, State, and local regulations for suspended particulates, sulfur oxides, and lead air pollutant emissions; wastewater effluent limitations; hazardous waste disposal requirements; and regulations directed at occupational safety and health. Compliance with some of these regulations may coincidentally decrease arsenic emissions or affect the manner in which arsenic emissions are discharged into the atmosphere, even though they were not necessarily developed with that objective. Each regulation must be evaluated to determine whether it will influence the engineering, environmental, energy, or economic conditions which will prevail in the industry when an arsenic NESHAP becomes effective. To specify a regulatory baseline from which to analyze the impacts of the proposed standards, it is necessary to review all of the major environmental, health, and safety regulations of the U.S. Environmental Protection Agency and the Occupational Safety and Health Administration which are applicable to the primary copper smelting industry.

Regulations to be examined in formulating the baseline conditions include the following:

- National Ambient Air Quality Standards under the Clean Air Act (CAA)
    - Sulfur Dioxide
    - Total Suspended Particulates
    - Lead
- Regulations (for Inorganic Arsenic) of the Occupational Safety and Health Administration (OSHA)
- Effluent limitation guidelines under the Clean Water Act (CWA)
- Hazardous waste disposal regulations under the Resource Conservation and Recovery Act (RCRA).
- Potential actions under the Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA)

For the purpose of relating the timings of the regulatory requirements cited above to the timing of an arsenic NESHAP, January 1986 is projected as the date of compliance with the arsenic NESHAP. The date is based upon a January 1984 promulgation date for the regulation and assumes that the full 2-year waiver of compliance available under the general provisions of 40 CFR 61 would be applied.

4.2.1.1 Clean Air Act. Under Section 109 of the Clean Air Act, EPA has set national ambient air quality standards (NAAQS) for certain "criteria" pollutants. The criteria pollutants emitted by primary copper smelters are sulfur dioxide, particulate matter, and lead. The NAAQS are to be met through the establishment of State implementation plans (SIP's) governing emission sources of these pollutants. The plans call for "combinations of emission limitations and other measures such that the total mix of these measures would result in the attainment and maintenance of air quality standards."[1]

The State of Washington SIP has designated that the Puget Sound Air Pollution Control Agency (PSAPCA) has jurisdiction over ASARCO-Tacoma smelter air emissions.[2] The PSAPCA Regulation I sets forth both ambient air quality standards and emission standards.

4.2.1.1.1 Sulfur dioxide. Sulfur dioxide ($SO_2$) emissions from the ASARCO-Tacoma smelter are covered by Sections 9.07(b) and 9.07(c) of PSAPCA Regulation I, which limit $SO_2$ emissions to a concentration of 2,000 ppm or less and limit the sulfur content of emissions to 10 percent or less of the sulfur contained in the process weight per hour.[3] ASARCO submitted an application to the PSAPCA Board of Directors for a variance from these regulations on December 5, 1975. The application was approved on February 19, 1976, granting a variance until December 31, 1980. An amended variance application was submitted on August 12, 1980, in order to reflect changes at the smelter, and to request an extension of the variance through 1982. A court decision issued on January 3, 1980, required compliance with the Washington State Environmental Policy Act (SEPA) for approval of a variance or variance extension. PSAPCA's Board of Directors therefore granted ASARCO an interim variance on April 10, 1980, which was in effect at the time of the August 12, 1980, request for an extension, and will be extended until the SEPA process is completed.

Section 119 of the Clean Air Act authorizes nonferrous smelter orders (NSO's) to delay the requirement for $SO_2$ emission controls. EPA issued final regulations for initial NSO's on June 24, 1980. PSAPCA has the primary responsibility for developing an NSO for ASARCO-Tacoma, with review of the Washington State Department of Ecology (DOE) and EPA. By a settlement agreement between EPA and ASARCO dated June 7, 1979, ASARCO has agreed to apply for an NSO for the ASARCO-Tacoma smelter, but completion of the application can be delayed until the final Supplementary Control System (SCS) regulations under Section 123 of the Clean Air Act are issued. ASARCO filed a letter of intent on August 4, 1980, to apply for an NSO.[2]

As of March 1983, SCS regulations have been prepared but not yet proposed. When final regulations are issued, PSAPCA and the Washington State DOE will determine the amount of sulfur emission reduction credit to be received by ASARCO for operating its supplementary control system. The outcome of this determination will be the deciding factor in whether the Puget Sound Intrastate Air Quality Control Region is classified as an attainment or nonattainment area for $SO_2$. If a nonattainment determination is made, ASARCO will be required to apply for an NSO. This would permit ASARCO to potentially delay compliance with the SIP for $SO_2$ emission limitations until January 1, 1993.[4]

In view of the uncertainties associated with the future course of the SCS and NSO programs with regard to ASARCO-Tacoma, it is difficult to predict the resulting $SO_2$ control requirements. However, since the Tacoma smelter currently utilizes control systems on process gas streams which generally reflect the best control technology for arsenic, the application of additional controls for $SO_2$ is expected to have a negligible impact on current arsenic emissions from process sources.

4.2.1.1.2 Total suspended particulates. TSP emissions from the ASARCO-Tacoma smelter are governed by PSAPCA Regulation I,[3] Sections 9.03(b), 9.09(c), and 9.09(d). Section 9.03(b) requires that the opacity of air emissions be 20 percent or less. Section 9.09(c) is a process weight regulation, while 9.09(d) is an absolute emission limit of 0.10 grains per standard cubic foot of exhaust gas.

ASARCO requested a variance from Section 9.03(b) as part of the variance application discussed for $SO_2$ emissions (see Section 5.2.1.1.1). The Tacoma smelter is in compliance with Sections 9.09(c) and 9.09(d).[5] Compliance with Section 9.09(c), the process weight regulation, requires the use of cold control devices on all process gas streams at the ASARCO-Tacoma smelter due to the high concentration of condensable constituents (such as arsenic) in the process offgases.

4.2.1.1.3 Lead. The national ambient air quality standard for lead of 1.5 $\mu g/m^3$ was promulgated on October 5, 1978 (40 CFR 51.80). PSAPCA has also promulgated an identical standard (Section 11.05 of Regulation I). As of March 1983, the State of Washington had not submitted an SIP for lead, though one was in the final stages of preparation.[5] Ambient monitoring in the vicinity of the Tacoma smelter has not revealed any violations of PSAPCA's AAQS for lead, however.[5] It is thus very unlikely that ASARCO-Tacoma will be required to implement additional emission control measures under the upcoming SIP.

4.2.1.2 Arsenic Regulation by the Occupational Safety and Health Administration. On May 5, 1978, the U.S. Occupational Safety and Health Administration promulgated standards for occupational exposure to inorganic arsenic.[5] The standards limit occupational exposure to 10 $\mu g/m^3$, averaged over any 8-hour period. The regulations require the use of engineering and work practice controls. In the event that engineering controls are not sufficient to reduce exposures to or below the standard, the engineering controls must be used to reduce exposures to the lowest level achievable and should be supplemented by the use of respirators and other necessary personal protective equipment. Primary copper smelters were required to monitor for violations and submit a compliance plan to OSHA by December 1, 1978.

The OSHA standard is in the form of an ambient concentration limit for the working environment at the smelter and does not specify the sources to be controlled nor the control technology which must be utilized. ASARCO has recently signed a tripartite agreement with OSHA and the United Steelworkers of America which "sets forth feasible controls and practices to protect the smelter employees from inorganic arsenic."[6] The terms of the agreement require ASARCO-Tacoma to maintain effective capture systems for fugitive emissions from calcine discharge and matte and slag tapping operations, and to install secondary hoods over the three large converters by July 1, 1984. The agreement does not, however, include technical specifications for the converter secondary hoods. Moreover, neither operating practices nor maintenance requirements for the secondary hoods are specified in the agreement. There are no provisions in the agreement for collection of captured fugitive emissions. Although the present analysis recognizes OSHA's requirement for capture of calcine discharge and matte and slag tapping fugitive emissions, capture of converter fugitive emissions is not assumed in the baseline case since the OSHA agreement is not specific as to the technology, performance criteria, or work practices to be employed for the capture of converter fugitive emissions.

4.2.1.3 Clean Water Act. Water pollution control regulations affect all areas of the primary copper industry, including mining, milling, smelting, refining, and acid plants. These regulations do not affect arsenic air emissions. The discussion of water pollution control regulations under the Clean Water Act is divided into those which are based on the best practicable control technology (BPT) requirements [Section 301(b)(1)(A)], and those which are based on the best available technology (BAT) and best conventional pollution control technology (BCT) [Section 301(b)(2)].

Potential water pollution originating from the ASARCO-Tacoma smelter will be regulated by EPA's proposed effluent limitations guidelines for nonferrous metals manufacturing.[7] Effluents from the ASARCO-Tacoma smelter will be covered by two subcategories: primary copper smelting and metallurgical acid plants. The proposed regulations for the primary copper smelting subcategory will amend promulgated BAT (40 FR 8523) to conform BAT to promulgated BPT (45 FR 44926), which is more stringent. The proposed metallurgical acid plant regulations set forth BAT effluent mass limitations for metallurgical acid plants, including copper smelter acid plants, based on promulgated BPT (45 FR 44926).

As of March 1983, the ASARCO-Tacoma smelter was in compliance with existing Federal water pollution regulations, and continued compliance was not expected to affect arsenic air emissions.[5]

4.2.1.4 Resource Conservation and Recovery Act (RCRA). On May 19, 1980, EPA promulgated regulations under the Resource Conservation and Recovery Act for the disposal of hazardous substances from metallurgical and other process industries (45 FR 33066). EPA identified the acid plant blowdown slurry/sludge resulting from the thickening of blowdown slurry at primary copper smelters as hazardous waste because of lead and cadmium content. As of early 1983, no further regulations which could be applied to ASARCO-Tacoma have been proposed or promulgated under RCRA. The present and projected impact of RCRA regulations on the ASARCO-Tacoma smelter, both in terms of arsenic air emissions and economic burden, is believed to be negligible.[8]

4.2.1.5 Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA). On December 30, 1982 (47 FR 58476), the Commencement Bay-Near Shore Tide Flats area, which includes the ASARCO-Tacoma smelter, was placed by EPA on the National Priorities List (NPL), a supplement to the National Oil and Hazardous Substances Contingency Plan (NCP), which was promulgated on July 16, 1982 (47 FR 31180), pursuant to Section 105 of the Comprehensive Environmental Response, Compensation, and Liability Act of 1980 (CERCLA), and Executive Order 12316. The NPL identifies priority releases of hazardous wastes, based on the assessments of State governments and EPA, for Fund-financed remedial action under CERCLA (also known as "Superfund").

As of early 1983, no determination has been made concerning possible actions at the Tacoma smelter under the Superfund program. It is therefore impossible to predict with any degree of certainty what economic impact, if any, Superfund activities will have on ASARCO-Tacoma. It is unlikely, however, that arsenic air emissions from the smelter will be affected by the Superfund program.[9]

4.2.1.6 Regulatory baseline summary. This section summarizes the pre-existing set of regulations that form the baseline from which the incremental impacts of the arsenic regulatory alternatives can be determined. Table 4-2 summarizes the existing regulations which affect arsenic air emissions at the ASARCO-Tacoma smelter. The effects of the regulatory baseline can be described as either technical, meaning they result in either a reduction in arsenic emissions or a

## Table 4-2. SUMMARY OF EXISTING REGULATIONS AFFECTING ARSENIC AIR EMISSIONS AT ASARCO-TACOMA

| Source | Capture | | Collection | |
|---|---|---|---|---|
| | Regulation | Requirement | Regulation | Requirement |
| Roaster Process TSP Emissions | Not Applicable | - | PSAPCA I[a] Section 9.03 (b) | Opacity of air emissions must be 20 percent or less. |
| | | | PSAPCA I Section 9.03(c) | Process weight regulation. Under typical operating conditions, allowable emissions at the roaster baghouse outlet are 52.7 lbs/hr. Corresponding As concentration is 0.01 gr/scf. |
| | | | PSAPCA I Section 9.03(d) | Maximum emission limit of 0.1 gr/scf. |
| Reverb Process TSP Emissions | Not Applicable | - | Same as for roaster | Under typical operations, 9.03(c) (process weight regulation) limits emissions from reverb ESP (#1) outlet to 52.1 lb/hr, corresponding to 0.04 gr/scf As. |
| Converter & Anode TSP Emissions[b] | Not Applicable | - | Same as for roaster | Under typical operations, 9.03(c) (process weight regulation) limits emissions from converter ESP outlet (#2, assumed full bypass of acid/$SO_2$ plants) to 51 lb/hr, corresponding to 0.02 gr/scf As. |
| Arsenic Plant TSP Emissions | Not Applicable | - | Same as for roaster | Under normal operating conditions, 9.03(c) (process weight regulation) limits emissions from the arsenic baghouse to 4 lb/hr, corresponding to 0.02 gr/scf As. |
| Calcine Discharge Fugitive Emissions | OSHA Compliance Plan | Maintain effective capture system | - | No collection requirement |
| Matte/Slag Tapping Fugitive Emissions | OSHA Compliance Plan | Maintain effective capture system | - | No collection requirement |
| Converter Operation Fugtive Emissions | OSHA Compliance Plan | Install secondary hoods[c] | - | No collection requirement |

[a] Puget Sound Air Pollution Control Agency. Regulation I. January 1983.

[b] Assuming worst case situation, i.e., complete bypass of acid/$SO_2$ plants by converter offgases. Offgases would be combined with anode furnace offgases and treated in the #2 ESP.

[c] No specification of technical, operating, or performance criteria. See Section 4.2.1.2.

TSP = Total Suspended Particulates.

change in the manner in which arsenic emissions are discharged to the atmosphere, or economic, in that the compliance costs of other regulations would affect the affordability of the regulatory alternatives. Such technical and economic effects are summarized in Table 4-3.

Under the regulatory baseline, the control technology required includes local hooding for capture of fugitive emissions from calcine discharge and matte and slag tapping; and a baghouse, or equivalent technology, operated at 110°C (230°F) or lower on all roaster, smelting furnace, and converter process gas streams.

### 4.2.2. Baseline Arsenic Emissions

A description of copper smelting processes, operations, and equipment at ASARCO-Tacoma was presented in Section 2.0. Potentially significant sources of arsenic emissions were identified. These include sources of both process and fugitive emissions from roasters, smelting furnaces, and converters. Specific fugitive emission sources identified include calcine discharge operations at multi-hearth roasters, smelting furnace matte and slag tapping operations, converter operations (charging, blowing, holding, and pouring), anode furnaces, dust handling and transfer, and the arsenic plant.

Baseline arsenic emissions, as presented in this section, refers to arsenic emissions from the ASARCO-Tacoma smelter under existing control. Existing process emitting equipment is described in Section 2.1.3, while a discussion of existing air pollution control equipment at ASARCO-Tacoma is contained in Section 3.2.

4.2.2.1 Baseline Arsenic Emission Calculations. Baseline arsenic emissions from individual process and fugitive sources at the Tacoma smelter were presented in Section 2.0 (Tables 2-4 and 2-11, respectively). These figures were derived from the arsenic material balance for ASARCO-Tacoma (Figure 2-6), employing the emission factors developed in Section 2.2.2 for fugitive emissions, and the estimated efficiencies of existing control devices based on test data presented in Section 3.3. In order to estimate total baseline arsenic emissions on an annual basis, a factor of 8,600 hours of operation per year was assumed. Based on this assumption, total potential process arsenic emissions are estimated to be 148 Mg/yr, while total potential fugitive arsenic

## Table 4-3. SUMMARY OF EFFECTS OF REGULATORY BASELINE ON ARSENIC REGULATORY ALTERNATIVES FOR THE ASARCO-TACOMA SMELTER

| Regulation | Technical Effects[a] | Economic effects |
|---|---|---|
| $SO_2$ - NAAQS | None likely | Unknown |
| TSP -NAAQS | None likely | None likely |
| Lead-NAAQS | None | None |
| OSHA | Capture and dispersion of roaster, smelting furnace, and converter fugitive emissions | Significant |
| CWA | None | None |
| RCRA | None | None |
| CERCLA | None | Unknown |

[a] Technical effects refer to alterations in the quantity or manner of arsenic emissions resulting from engineering practices utilized for compliance with baseline regulations.

emissions amount to 134 Mg/yr, yielding a total estimated potential baseline arsenic emission rate of 282 Mg/yr.

## 4.3 REGULATORY ALTERNATIVES

Alternative techniques for the control of both process and fugitive arsenic emission sources were identified and discussed in Section 3.0. A summary of the candidate systems selected as bases for regulatory alternatives and the rationale for this selection follows.

### 4.3.1 Process Emission Control Techniques

Effective techniques for the control of arsenic emissions contained in the process gases produced during roasting, smelting, and converting were described in Section 3.1.1. The control technique identified as best consisted of the application of a baghouse or an equivalent control device, such as an electrostatic precipitator preceded by gas stream cooling, or a high energy venturi scrubber. Sulfuric acid plants also result in effective arsenic control due to the need to incorporate several of the above devices for gas precleaning purposes. Existing process emission controls at ASARCO-Tacoma are described in Section 3.2.

4.3.1.1 Roaster Process Emission Controls. Roaster offgases at the Tacoma smelter are cooled to a temperature less than 120°C (250°F) and treated in a baghouse system. Simultaneous inlet and outlet arsenic emission measurements were performed by EPA across the baghouse serving the roasters at ASARCO-Tacoma, as described in Section 3.3.1.1.1. The average arsenic removal efficiency for this unit, as indicated by the test results, was 99.7 percent.

4.3.1.2 Reverberatory Furnace Process Emission Controls. Furnace offgases at the ASARCO-Tacoma smelter are cooled to a temperature of 132°C (270°F) before treatment in an electrostatic precipitator system. EPA performed outlet arsenic emission measurements at the electrostatic precipitator, as described in Section 3.3.1.2.1. The test results, combined with an estimate of inlet arsenic loading from the arsenic material balance (Section 2.2.1) yield an estimated arsenic removal efficiency of 98 percent for the reverberatory furnace electrostatic precipitator.

4.3.1.3 Converter Process Emission Controls. Converter offgases at ASARCO-Tacoma are treated in the gas cleaning circuit associated with either the liquid $SO_2$ plant or a single-contact sulfuric acid plant for particulate removal, as described in Section 3.2.1. The gas cleaning circuits of the $SO_2$ control devices consist of a water spray chamber, electrostatic precipitator, scrubbers, and mist precipitators in series. Occasionally, when converter offgas flows exceed the capacity of the $SO_2$ control system, some amount of the offgases bypasses the system and is treated in a "cold" (150 to 200°F) electrostatic precipitator for particulate matter removal before discharge through the main stack.

Section 3.3.1.4 describes EPA's evaluation of the arsenic removal performance of various acid plant systems at domestic copper smelters. Based on these findings, an arsenic removal efficiency of 99 percent was ascribed to the $SO_2$ control devices at ASARCO-Tacoma.

### 4.3.2 Fugitive Emission Control Techniques

A variety of techniques is available for controlling arsenic emissions from fugitive sources located at copper smelters. Section 3.1.2 presents background information on fugitive arsenic emission control techniques. Reduction of fugitive arsenic emissions first requires capturing the emissions. Once captured, the emissions may be vented directly to a control device for collection or combined with process gases prior to collection in a control device.

4.3.2.1 Roaster Fugitive Emission Controls. Of the systems examined for the control of fugitive emissions from multi-hearth roaster calcine transfer operations, only the use of a larry car shed or tunnel coupled with local ventilation applied at the calcine hopper-to-larry car transfer point was judged to be effective. Details of the system, which is presently used at the ASARCO-Tacoma smelter, were described in Section 3.2.2. The performance evaluation was detailed in Section 3.3.2.1. Visual observations were made across the shed opening. No visible emissions were observed at any time during the observation period. Based on these observations, the system was judged to achieve a capture efficiency of 90 percent or better. The captured fugitive emissions are combined with the roaster process offgases and treated in the roaster baghouse for particulate removal.

4.3.2.2 Reverberatory Furnace Fugitive Emission Controls. Local ventilation hoods are used at ASARCO-Tacoma to capture fugitive arsenic emissions from furnace matte and slag tapping operations, as described in Section 3.2.2. Fixed hoods are placed over tapping ports. At ladle and pot filling points, retractable hoods are used. Based on visual observations, detailed in Section 3.3.2.1, capture efficiencies in excess of 90 percent are achievable using local ventilation hoods. Captured emissions from both systems are combined with anode furnace offgases and treated in an electrostatic precipitator at 66 to 93°C (150 to 200°F) before being discharged through the main stack.

4.3.2.3 Converter Fugitive Emission Controls. Fugitive arsenic emissions from converters can be captured by using local ventilation techniques. Local ventilation techniques evaluated consisted of using mechanical hoods, air curtain systems, or a combination of these two (refer to Section 3.1.2.7). Capture efficiencies for fixed mechanical hoods are estimated to be 80 percent or less.

The prototype air curtain secondary hood system recently installed on converter No. 4 at ASARCO-Tacoma is described in Section 3.1.2.7.2. Captured emissions are presently routed to the anode furnace and matte and slag tapping electrostatic precipitator for particulate matter removal. In a test program at the smelter, the overall average capture efficiency of the air curtain system was determined to be about 95 percent (refer to Section 3.3.2.2.1).

The existing prototype air curtain secondary hood on converter No. 4 is the first of three such installations planned by ASARCO at its Tacoma plant. As such, the prototype system has been used to collect performance data and refine design and operation parameters. Since this system is in what may be called a "shakedown" or optimization phase, it is not included in the baseline. There is at present no Federally enforceable regulation which requires the installation and continued operation of a device such as the prototype air curtain hood system in place on the No. 4 converter.

Electrostatic precipitators, baghouses, or wet scrubbers can be used to collect arsenic emissions (refer to Section 3.1.1). For the regulatory alternatives, any of the above devices, well-maintained and operated at 120°C (250°F) or less, qualifies for control of the affected facilities. This selection is based on source test data which show that at least 96 percent arsenic collection efficiency can be achieved by such devices.

For the purpose of the regulatory alternatives, the fugitive arsenic emission control technique considered for converter operations is emission capture by using an air curtain secondary hood, and collection of the captured emissions by a well-maintained cold particulate control device as described above.

### 4.3.3 Regulatory Alternative I

Regulatory Alternative I represents no regulatory action (baseline) and would require no additional controls beyond existing process and fugitive emission controls as described in Section 3.0, for ASARCO-Tacoma. It should be noted that although converter No. 4 is presently equipped with a prototype air curtain secondary hood vented to a cold electrostatic precipitator, Regulatory Alternative I gives no credit for its existence.

### 4.3.4 Regulatory Alternative II

Regulatory Alternative II includes all controls specified by Regulatory Alternative I, plus air curtain secondary hoods on three converters for capture of converter fugitive emissions, and collection of captured emissions using a baghouse or equivalent technology. For the purposes of cost analysis in Section 6.0, an existing ESP ("#2") is assumed to be used for collection of captured converter fugitive emissions.

### 4.3.5 Regulatory Alternative III

Regulatory Alternative III would require that arsenic emissions from the ASARCO-Tacoma smelter be reduced to zero. Accomplishment of this alternative would require the ASARCO-Tacoma smelter to process ores which were virtually free of arsenic content. Implementation of Regulatory Alternative III would therefore result in closure of the ASARCO-Tacoma smelter.

## 4.5 REFERENCES

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2. Puget Sound Air Pollution Control Agency. Final Environmental Impact Statement for ASARCO, Incorporated (Summary), September 1980.

3. Puget Sound Air Pollution Control Agency. Regulation I. January 1983.

4. Telecon. Whaley, G., Pacific Environmental Services, with Hooper, M., EPA Region X. Federal Environmental Regulations Applicable to the ASARCO-Tacoma Smelter. March 24, 1983.

5. U.S. Department of Labor, Occupational Safety and Health Administration. General Industry Standards, OSHA Safety and Health Standards (29 CFR 1910), and OSHA 2206, Revised November 7, 1978.

6. Occupational Safety and Health Administration. Engineering Assessment and Proposed Compliance Plan for ASARCO Inc. Tacoma, Washington. January 1982.

7. U.S. Environmental Protection Agency. Nonferrous Metals Manufacturing Point Source Category; Effluent Limitations Guidelines, Pretreatment Standards, and New Source Performance Standards. Federal Register, Vol. 48. February 17, 1983. p. 7032.

8. Telecon. Whaley, G., Pacific Environmental Services, with Hofer, G., EPA Region X. Federal Environmental Regulations Applicable to the ASARCO-Tacoma Smelter. March 30, 1983.

9. Telecon. Whaley, G., Pacific Environmental Services, with Davoli, D., EPA Region X. Federal Environmental Regulations Applicable to the ASARCO-Tacoma Smelter. March 28, 1983.

# 5.0 ENVIRONMENTAL IMPACTS

## 5.1 INTRODUCTION

The environmental impacts on air, energy consumption, solid waste, and water associated with the regulatory alternatives for the control of arsenic emissions from high arsenic throughput smelters are presented in this chapter. The ASARCO smelter in Tacoma, Washington, is the only U.S. smelter in the high arsenic throughput copper smelter category. Therefore, the impacts presented in Section 5.0 represent those for the ASARCO-Tacoma smelter. The purpose of this analysis is to determine the incremental change in air pollution, water pollution, solid waste, and energy impacts of the regulatory alternatives over the baseline control level. The baseline control level reflects existing levels of control of arsenic emissions at the ASARCO-Tacoma smelter. This level is represented by Regulatory Alternative I.

Section 5.1 addresses the air pollution impacts of implementing each of the regulatory alternatives. Water pollution, solid waste, and energy impacts for the regulatory alternatives are addressed in Sections 5.2, 5.3, and 5.4, respectively.

## 5.2 AIR POLLUTION IMPACTS OF REGULATORY ALTERNATIVES

The air pollution impact associated with each of the regulatory alternatives considered is presented in this section. Incremental and cumulative arsenic emission reductions are discussed for the ASARCO-Tacoma smelter. The emission estimates are obtained based on the application of capture and collection control systems selected in Section 3.3 as the bases for the regulatory alternatives.

### 5.2.1 Baseline Emissions

The baseline regulatory alternative for ASARCO-Tacoma includes effective process controls on roasters, furnaces, converters, and the arsenic plant, and local ventilation capture systems for fugitive emissions from roasters and furnaces. Controls for process emissions at the ASARCO-Tacoma smelter include exhaust gas cooling by tempering air and baghouse collection for the roasters; gas cooling by air infiltration and evaporative cooling followed by electrostatic precipitators for the reverberatory furnace; a baghouse for arsenic plant emissions; and electrostatic precipitators and scrubbers for particulate removal prior to treatment in the sulfuric acid plant or $SO_2$ plant for the converters. Process emissions from the anode furnace are captured and vented to an electrostatic precipitator. Collection efficiencies for these process control systems and the resultant baseline process arsenic emission rates are summarized in Table 5-1. Fugitive emissions from the roaster calcine discharge operation are combined with the roaster process gases and treated in a baghouse for particulate removal. Fugitive emissions from furnace matte tapping and slag tapping operations are captured by local ventilation, combined with reverberatory furnace offgases, and collected in an electrostatic precipitator. Fugitive arsenic emissions from converter and anode furnace operations are not controlled. Fugitive arsenic emissions from the arsenic plant are controlled by means of dust control work practices. Fugitive arsenic emissions from the flue dust handling system are controlled by means of dust-tight transfer systems. Capture and collection efficiencies for fugitive emissions control systems and the estimated baseline fugitive arsenic emission rates are summarized in Table 5-2.

Baseline arsenic emissions, presented in Tables 5-1 and 5-2, were calculated by determining the potential and controlled arsenic emission rate for each process and fugitive emission source. Potential arsenic emission rates were determined based on the arsenic material balance for the smelter (Figure 2-6) and emission factors presented in Section 2.2. Controlled arsenic emission rates were determined by application of the estimated capture and collection efficiencies of the baseline emissions control systems described above to each process and fugitive emissions source. Capture and collection efficiencies of the control equipment are based on control equipment performance test results presented in Section 3.1. Potential arsenic emissions were calculated based on an arsenic feed rate of 941 kg/hr to the smelter.[1] This arsenic enters the smelter in the copper ore concentrate and other copper-bearing materials processed

Table 5-1. PROCESS EMISSION SOURCES, CONTROL EFFICIENCIES, AND ARSENIC EMISSION RATES[a]

| Process Emission Source | Potential Arsenic Emissions (kg/hr) | Control Device | Control Efficiency (Percent) | Baseline Arsenic Emission Rate (kg/hr) |
|---|---|---|---|---|
| Roasters | 255 | Baghouse | 99.8 | 0.4 |
| Reverberatory Furnaces | 608 | ESP | 98.0 | 9.5[b] |
| Converters[c] | 94 | ESP<br>ESP/Acid Plant<br>ESP/$SO_2$ Plant | 96.0<br>99.9<br>99.9 | 0.04[d] |
| Arsenic Plant | 373 | Baghouse | 98 | 7.3 |
| Anode Furnace | 0.4 | ESP | 96 | 0.02 |
| Total | 1,330 | | | 17.3 |

[a] From Reference 1.

[b] Of the potential arsenic emissions that leave the process, a portion is collected from the flue gas handling system as dust fallout prior to entering the control device indicated.

[c] Process offgases from the converters are separated into three separate streams, each directed to the control device(s) shown.

[d] The arsenic emission rate shown represents the sum of controlled emissions from the three control systems through which the converter offgases pass.

## Table 5-2. FUGITIVE EMISSION FACTORS, CAPTURE AND COLLECTION EFFICIENCIES USED TO DETERMINE FUGITIVE ARSENIC EMISSIONS

| Fugitive Emissions Source | Fugitive Emissions Factor | Potential Arsenic Emissions (kg/hr) | Efficiency of Capture System (Percent) | Efficiency of Collection System (Percent) | Overall Control Efficiency (Percent) | Baseline Arsenic Emission Rate (kg/hr) |
|---|---|---|---|---|---|---|
| Roaster Calcine Transfer | 0.03% of Arsenic in Calcine | 0.3 | 90 | 99.8[a] | 89.8 | 0.03 |
| Furnace Slag Tapping | 0.12% of Arsenic in Slag from Reverberatory Furnace | 0.25 | 90 | 96 | 86.4 | 0.5 |
| Furnace Matte Tapping | 2.5% of Arsenic in Matte from Reverberatory Furnace | 3.8 | | | | |
| Converter Slag Return | 0.03% of Arsenic in Converter Slag | 0.01 | $0^b$ | $0^b$ | $0^b$ | 0.01 |
| Converter Operations | 15% of Arsenic in Converter Off-Gases | 14.0 | $0^b$ | $0^b$ | $0^b$ | 14.0 |
| Anode Furnace Operations | 20% of Total Arsenic Emissions from the Anode Furnace[c] | 0.08 | $0^c$ | $0^c$ | $0^c$ | 0.08 |
| Miscellaneous (Flue Dust Handling System) | 0.1% of Arsenic in all Dust Collected | 2.7 | d | d | 90 | 0.3 |
| Arsenic Building Operations | 0.5% of Arsenic to Arsenic Plant | 6.4 | e | e | 90 | 0.6 |
| Total | | 27.5 | | | | 15.5 |

[a] Captured fugitive emissions are ducted to the roaster baghouse.

[b] No controls in baseline emission case.

[c] Assumes that 11 percent of the arsenic input to the anode furnace represents total (process plus fugitive) potential anode furnace emissions. Twenty percent of these emissions represents potential fugitive emissions. These fugitive emissions are not controlled.

[d] Flue dust transfer, handling, and conveying emissions are estimated to be 90 percent controlled by means of dust-tight transfer systems.

[e] Arsenic building emissions are estimated to be 90 percent controlled by means of dust control work practices.

at the smelter. As shown in Tables 5-1 and 5-2 for the baseline case, the total process arsenic emission rate is 17.3 kg/hr and the total fugitive arsenic emission rate is 15.5 kg/hr. Annual arsenic emissions are determined using these emission rates and assuming 8,600 hours of smelter operation per year. The resultant annual baseline arsenic emissions for the ASARCO-Tacoma smelter are 282 Mg/yr.

### 5.2.2 Arsenic Emission Reductions Under the Regulatory Alternatives

Table 5-3 presents the estimates of annual arsenic emissions and the percentage emission reductions achievable through application of the different regulatory alternatives being considered for the ASARCO-Tacoma smelter. These estimates include both total process emissions and total fugitive emissions. Annual estimates of arsenic emissions were determined assuming 8,600 hours of operation per year. The reduction in arsenic emissions associated with each regulatory alternative is obtained by subtracting the emission rate for the regulatory alternative from the baseline emission rate.

Arsenic emissions under Regulatory Alternative II were calculated by applying the additional controls selected for the alternative to the baseline case. The additional controls include the installation of an air curtain secondary hood to each of the three operating converters at the smelter to capture the fugitive arsenic emissions discharged by the converters. Collection of the captured fugitive arsenic emissions will be achieved by a refurbished, existing electrostatic precipitator (ESP). The air curtain secondary hood is 95 percent efficient, and the ESP collection system is 96 percent efficient. Arsenic emissions under Regulatory Alternative II are 172 Mg/yr. This represents a reduction of 110 Mg/yr, or 39 percent, from the baseline arsenic emission level.

Regulatory Alternative III requires that all arsenic emissions from the ASARCO-Tacoma smelter be reduced to zero. This represents a reduction of 100 percent over the baseline arsenic emission level.

## 5.3 ENERGY IMPACTS OF THE REGULATORY ALTERNATIVES

Table 5-4 summarizes the incremental energy requirements of the regulatory alternatives over the baseline case for the ASARCO-Tacoma smelter. The additional energy requirements for Alternative II are

## Table 5-3. ARSENIC EMISSIONS AND EMISSION REDUCTIONS AT ASARCO-TACOMA UNDER THE REGULATORY ALTERNATIVES

| Regulatory Alternative | Arsenic Emissions (Mg/yr) | Arsenic Emission Reductions from Baseline (Mg/yr) | Arsenic Emission Reductions from Baseline (Percent) |
|---|---|---|---|
| Alternative I (Baseline) | 282 | -- | -- |
| Alternative II | 172 | 110 | 39 |
| Alternative III | 0 | 282 | 100 |

## Table 5-4. ANNUAL ENERGY REQUIRED BY AIR POLLUTION CONTROL EQUIPMENT AT ASARCO-TACOMA UNDER THE REGULATORY ALTERNATIVES

| Regulatory Alternative | Incremental Annual Energy Requirements from Baseline ($10^6$ kWh) |
|---|---|
| Baseline | -- |
| Alternative II | 15 |
| Alternative III | 0 |

calculated based on the total electrical energy required by the air curtain secondary hood and ESP collection system for control of converter fugitive emissions. The electrical energy requirements are determined based on the total gas flow to be handled by the air curtain secondary hood and ESP, assuming 8,600 hours of operation per year. The energy requirements for Alternative II include 10 percent additional energy for miscellaneous equipment. At ASARCO-Tacoma, Regulatory Alternative II requires an additional 15 million kWh of electricity over the baseline case. Total smelter electrical energy requirements were estimated by assuming that the smelter requires $49 \times 10^6$ Btu (thermal) per ton of anode copper produced.[2] The ASARCO smelter has a reported copper production capacity of 91,000 Mg/year (100,000 tons/year).[3] Assuming a power plant efficiency of 35 percent and 8,600 hours of operation per year, the total electrical energy requirement of the ASARCO smelter was estimated to be $3 \times 10^{10}$ kWh. Compared to this total, the additional energy requirements of Regulatory Alternative II are negligible. Regulatory Alternative III has no energy requirements.

## 5.4 SOLID WASTE IMPACTS OF THE REGULATORY ALTERNATIVES

Arsenic-bearing dust is collected when smelting process particulate emissions are controlled with baghouses or electrostatic precipitators. Often, this dust contains recoverable copper or other salable materials. The collected dust is recycled to the process or reclaimed elsewhere. Thus, only a portion of the material collected by air pollution control equipment becomes solid wastes. At ASARCO-Tacoma, arsenic-containing dust collected from the roaster baghouse and the reverberatory furnace electrostatic precipitator is used as input material to the arsenic plant. A portion of the dust collected in the converter electrostatic precipitators is sent to the arsenic plant with the balance sent to the fine ore bin. From the fine ore bin, the dust is recycled back to the process. Arsenic-containing wastes are present in the acid plant waste. Acid plant waste is usually in the form of a slurry. State regulations require settling of this slurry in a concrete pit. The clarified slurry is transferred to a lined lagoon for further settling. From the lagoon the materials may be dredged and recycled to the process.[4]

The total amount of solid waste generated under the baseline control case at ASARCO-Tacoma can be determined by assuming that 50 percent of the concentrate fed to the smelter becomes solid waste, 25 percent is sulfur removed as $SO_2$, and the remaining 25 percent becomes blister copper. Given an annual maximum concentrate feed rate of 363,000 Mg/yr (400,000 tons/yr)[3], solid wastes (including slag) generated by the smelter are approximately 181,500 Mg/yr (200,000 tons/yr).

Incremental solid wastes generated under Alternative II were calculated by assuming that the air curtain secondary hood system will capture 95 percent of converter fugitive emissions and the ESP will have a collection efficiency of 96 percent. This represents an overall control efficiency of 91.2 percent for the converter fugitive emissions control system. The fugitive emissions collected under Alternative II are assumed to have no value and will, therefore, be disposed of as waste. Assuming that the arsenic content of the fugitive emissions collected is 1 percent, the incremental solid waste impact under Alternative II is calculated at 11,000 Mg/yr. Under Alternative III, solid wastes generated by the smelter would reduce to zero. Table 5-5 summarizes the incremental solid waste captured for the regulatory alternatives at ASARCO-Tacoma.

In comparison with the amount of solid waste generated by the smelter under the baseline case, the additional amount of solid waste generated under Regulatory Alternative II is negligible.

## 5.5 WATER POLLUTION IMPACTS OF THE REGULATORY ALTERNATIVES

The control systems for the regulatory alternatives are dry systems; consequently, there will be no incremental increase in water discharges. If scrubbers are used, increases in wastewater discharges result if the arsenic-containing dusts are disposed of along with the acid plant slurry. Even if scrubbers are used, no adverse water pollution impact is anticipated, because the additional wastewater discharges through use of scrubbers would be treated within existing smelter water pollution control systems installed to meet existing State and Federal regulations.

Table 5-5. SOLID WASTES GENERATED BY AIR POLLUTION CONTROL EQUIPMENT AT ASARCO-TACOMA BY REGULATORY ALTERNATIVE[a]

| Regulatory Alternative | Incremental Solid Wastes Generated by Air Pollution Control Equipment from Baseline (1,000 Mg/yr)[b] |
|---|---|
| Baseline | -- |
| Alternative II | 11.0 |
| Alternative III | 0 |

[a] Based on 8,600 hours per year.

[b] Represents total solid waste including slag disposed of by the smelter.

## 5.6 REFERENCES

1. Letter and Attachments from Mr. M.O. Varner, ASARCO, Incorporated, to J.R. Farmer, U.S. Environmental Protection Agency. March 16, 1983. Response to Section 114 Information Request.

2. Environmental Impact Statement for ASARCO, Incorporated. Variance from PSAPCA Regulation I, Sections 9.03(b), 9.07(b), and 9.07(c). Summary. Final Report. Puget Sound Air Pollution Control Agency (PSAPCA). September 1981. p. 36.

3. Preliminary Study of Sources of Inorganic Arsenic. U.S. Environmental Protection Agency. Research Triangle Park, North Carolina. Report No. EPA 68-02-3513. August 1982. p. 21.

4. Calspan Corporation. Assessment of Industrial Hazardous Waste Practices in the Metals Smelting and Refining Industry. Volume II, Primary and Secondary Nonferrous Smelting and Refining. PB 276170. April 1971.

# 6.0 COSTS

As discussed in earlier sections, only ASARCO-Tacoma processes high arsenic content feed. None of the other existing smelters processes high arsenic content feed nor is anticipated to do so in the future. In addition, no new smelters are expected to be built in the next 5 years. Therefore, the cost analysis developed in this section applies only to the ASARCO-Tacoma smelter.

This section provides estimates of capital and annualized costs to control the arsenic emissions from the converter operations at the ASARCO-Tacoma smelter. These cost estimates provide the basis for the implementation of each of the regulatory alternatives identified in Section 4.0 for the control of arsenic emissions from the smelter. As discussed in earlier sections, all the process emission sources and fugitive emission sources other than from converter operations at the smelter are effectively controlled; therefore, the regulatory alternatives discussed in Section 4.0 do not include additional controls for these sources. As a result, no additional expenditure would be required under the regulatory alternatives to control emissions from any source other than the fugitive emissions from the converter operations. Consequently, the cost analysis will be limited to developing cost estimates for the control of fugitive emissions from converter operations.

## 6.1 EXISTING FACILITY

The ASARCO-Tacoma smelter operates three Pierce-Smith converters. The capital and annualized costs for controlling fugitive emissions from all three converters are developed in this section. These costs are based on site-specific parameters.

### 6.1.1 Control System

As noted in Section 4.0, the control technique selected for the control of fugitive arsenic emissions from converter operations consists of applying an air curtain secondary hood for capture followed by an effective particulate matter control device (96 percent efficient) for collection. The air curtain secondary hood systems to be installed are essentially identical to the prototype secondary hood presently installed on the No. 4 converter at the smelter. ASARCO intends to use an existing ESP at the smelter for the collection of captured fugitive emissions from the converter operations. Thus, it is assumed that no additional capital expenditure will be required for collection of these captured fugitive emissions. Therefore, the cost analysis addresses both the capital and operating costs for equipment to capture the fugitive emissions, and only the operating costs for fugitive emission collection equipment.

### 6.1.2 Cost Parameters

The cost estimates for the converter capture systems are developed based on the design parameters summarized in Table 6-1. The parameters were obtained from ASARCO.[1]

The ASARCO design consists of an air curtain and hood enclosure for each converter. The top, back, and sides of the hood are fully enclosed. The top and sides are extended into the converter aisle area. Converter charging and skimming can be performed by bringing a ladle into and out of the enclosure through the front end opening. The captured gases from the enclosure are evacuated through a take-off located at one end on the top of the enclosure to a dust chamber and then to a main fan. An air curtain screen is located at the top of the hood enclosure directly opposite the exhaust take-off for the enclosure. Each air curtain has its own fan. A common duct from the main fan exhausts the gases to an existing flue, then to an ESP.

Exhaust rates through the enclosure are controlled based on the number of active converters and their modes of operation. (See Section 3.1.2.7 for a detailed description of the system.) Exhaust fan requirements assume that, at most, only two converters will be in operation at a time. Therefore, the main fan will handle gases from, at most, two hood enclosures. The main fan is designed for 110 $m^3/s$ (230,000 acfm) to handle the maximum volume of gases from a worst-case condition of one converter in roll-in and one in roll-out mode. The fan is specified for 930 kW (1,200 hp) and 5.5 kPa (22 in. water) pressure requirement. This fan power requirement is sufficient to

Table 6-1. DESIGN PARAMETERS FOR THE AIR CURTAIN SECONDARY HOOD AND ESP FOR ASARCO-TACOMA[a]

| Parameter | Value |
|---|---|
| **Air curtain** | |
| Gas flow rate: | |
| During converter roll-in[b], $m^3/s$ (acfm) | 5.2 (11,000) |
| During converter roll-out[c], $m^3/s$ (acfm) | 8.5 (18,000) |
| System pressure drop, kPa (in. water) | 7.5 (30) |
| Fan power requirements, kW (hp)[d] | 224 (300) |
| **Secondary hood system** | |
| Gas flow rate: | |
| During converter roll-in[b], $m^3/s$ (acfm) | 33 (70,000) |
| During converter roll-out[c], $m^3/s$ (acfm) | 57 (120,000) |
| Fan capacity, $m^3/s$ (acfm)[e] | 109 (230,000) |
| Pressure drop, kPa (in. water)[f] | 5.5 (22) |
| Fan power requirements, kW (hp)[f] | 930 (1,250) |
| **ESP** | |
| Maximum gas flow rate, $m^3/s$ (acfm) | 109 (230,000) |

[a] Each of the three existing converters will be equipped with an air curtain secondary hood capture system.

[b] Converter blowing or holding.

[c] Converter charging or skimming.

[d] Total for two air curtains. Each air curtain has a 112 kW (150 hp) fan.

[e] Based on one converter in roll-in and one converter in roll-out mode.

[f] To handle pressure drop across the entire system including hood enclosure, ESP, ducting, and stack.

force the gases through the hood enclosures, ESP, and stack to the atmosphere. Duct requirements include (1) a total of 229 m (750 ft) of 1.52 m (60 in.) diameter duct to convey emissions from the three secondary hoods to a dust chamber and then to the main exhaust fan, and (2) 91 m (300 ft) of 2.18 m (86 in.) diameter duct to exhaust the gases from the fan to the existing flue system. The air curtain fan requirement for each of the three converters to achieve the needed air jet pressure was estimated at 8.5 $m^3/s$ (18,000 acfm) maximum. The fan power requirement for each air curtain was estimated at 112 kW (150 hp).

### 6.1.3 Capital and Annualized Costs

Capital cost is estimated for the air curtain secondary hood. Annualized cost is estimated for the air curtain secondary hood and ESP. The capital cost includes all the cost items necessary to design, purchase, and install the capture system. It includes the cost of air curtain fans, motors, hood structural material, main exhaust fan, and ductwork; direct installation charges including foundation and other direct costs such as electrical, instrumentation, and controls; and indirect costs for engineering services, procurement, taxes, fees, and contingency. All costs are in December 1982 dollars.

The annualized cost of a control system is the annual cost to the plant to own and operate that control system. The annualized cost includes direct operating costs such as utilities, maintenance, and operating labor; and indirect operating costs or capital-related charges such as depreciation, interest, administrative overhead, property taxes, and insurance.

Capital cost estimates are developed based on the cost data provided to EPA by ASARCO.[1] The ASARCO cost data contain January 1982 cost estimates for the air curtain secondary hood capture system with doors for the three converters at the Tacoma smelter. Direct capital costs are based on actual estimates, and indirect capital costs are based on percentages of the estimated equipment costs.

Tables 6-2 and 6-3 present the capital and annualized cost estimates for air curtain secondary hood systems at the ASARCO-Tacoma smelter, in December 1982 dollars.

Table 6-2. ESTIMATED CAPITAL COSTS FOR THE AIR CURTAIN SECONDARY HOOD SYSTEM AT ASARCO-TACOMA
(December 1982 dollars)

| Item | Capital Cost, $ |
|---|---|
| Air curtains | 375,000 |
| Hoods | 591,000 |
| Duct work | 2,444,000 |
| Electrical system | 59,000 |
| Total | 3,469,000 |

Table 6-3. ESTIMATED ANNUAL OPERATING COSTS FOR THE AIR CURTAIN SECONDARY HOOD AND ESP SYSTEMS FOR ASARCO-TACOMA (December 1982 dollars)

| Item | Cost, \$/yr |
|---|---|
| Direct costs | |
| Operating and maintenance labor | 58,200 |
| Maintenance material | 29,800 |
| Utilities | 859,800[a] |
| Indirect costs | |
| Payroll overhead | 35,000 |
| Operating supplies | 6,000 |
| Administration overhead | 25,700 |
| Taxes and insurance | 69,400 |
| Capital recovery | 407,600[b] |
| Total | 1,491,500 |

[a] $94,000 is for electricity input to the ESP, $585,800 for electricity input to air curtain secondary hoods, and $180,000 for electricity input for miscellaneous purposes.

[b] For the air curtain secondary hood system only.

The capital cost estimate ($3.47 million) was obtained by first calculating the January 1982 cost of air curtain secondary hood systems without doors (from the ASARCO cost estimate for air curtain secondary hood systems with doors), and then applying an escalation factor of 1.03 to update the January 1982 cost to December 1982. This cost estimate of $3.47 million is about 25 percent less than that estimated by ASARCO for air curtain secondary hoods with doors. Annualized costs were developed using the cost bases summarized in Table 6-4.

### 6.1.4 Costs of Regulatory Alternatives

This section presents the costs associated with the implementation of the regulatory alternatives defined in Section 4.0. Regulatory Alternative I represents the baseline and includes all the existing process and fugitive emission controls. Regulatory Alternative II includes the baseline and the converter fugitive emission capture and collection system. Regulatory Alternative III requires zero emissions. Since neither Regulatory Alternative I (baseline) nor Regulatory Alternative III requires the installation of additional emission control equipment at the smelter, no cost is estimated for either alternative. Therefore, the only additional control cost involved is that associated with Regulatory Alternative II which requires the installation of an air curtain secondary hood on all three operating converters at the smelter.

As presented in Table 6-3, the annualized control cost for implementing Regulatory Alternative II is estimated at $1.49 million. Assuming an annual smelting capacity of 90,720 Mg/yr (100,000 tons/yr) of copper is achieved, the additional cost of implementing Regulatory Alternative II is $16.4/Mg ($14.9/ton) of copper.

### 6.1.5 Cost-Effectiveness

Cost-effectiveness is defined as incremental annual costs in dollars per unit of pollutant removed over baseline. As discussed in Section 5.0, the arsenic emission reduction achievable under Regulatory Alternative II is estimated at 12.8 kg/hr (28.2 lb/hr), or 110 Mg/yr (121 tons/yr). Therefore, the incremental cost-effectiveness associated with implementing Regulatory Alternative II over baseline is $13,600/Mg ($12,300/ton) of arsenic removed.

Table 6-4. COST BASES USED IN ESTIMATING ANNUAL OPERATING COSTS OF THE AIR CURTAIN SECONDARY HOOD AND ESP SYSTEM FOR ASARCO, TACOMA[a]

| Item | Cost bases | Comment |
|---|---|---|
| Direct costs | | |
| Operating and maintenance labor[b] | 2 labor-hours operating labor and 15 percent supervision per shift, 2 labor-hours maintenance labor and 20 percent supervision per shift, and $11.53/labor-hour | The labor-hour unit cost of $11.53/hour is obtained from Reference 2. |
| Maintenance material | 100 percent of maintenance labor | Based on U.S. Bureau of Mines methodology |
| Utilities | 230,000 cfm gas flow at 22 in. water pressure, 0.0015 kW/ft$^2$ of ESP collection area, 10 percent additional energy for miscellaneous requirements, and $0.059 per kWh. | The unit power cost of $0.059/kWh is obtained from Reference 3. |
| Indirect costs | | |
| Payroll overhead | 60 percent of payroll | |
| Operating supplies | 20 percent of total maintenance cost | Based on U.S. Bureau of Mines methodology |
| Administrative overhead | 40 percent of total direct labor and operating supplies | |
| Taxes and Insurance | 2 percent of total capital cost | |
| Capital recovery | 20 years life and 10 percent interest rate for both capture and collection equipment | Equipment life is obtained from Reference 4. |

[a] System is assumed to operate 8,600 hours in a year.

[b] Estimate based on 1 labor-hour each of operating and maintenance labor for the air curtain secondary hood system and for the ESP.

## 6.2 REFERENCES

1. ASARCO's Central Engineering Department. ASARCO Incorporated Converter Secondary Hooding, Tacoma Plant. Salt Lake City, Utah. January 22, 1981.

2. Survey of Current Business, U.S. Department Labor, Washington, DC. December 1982.

3. Monthly Energy Review. U.S. Department of Commerce. DOE/EIA 033583/01. Washington, DC. January 1983.

4. GARD, Inc. Capital and Annual Operating Costs of Selected Air Pollution Control Systems. U.S. Environmental Protection Agency. EPA Contract No. 68-02-2812. December 1978.

# 7.0 ECONOMIC IMPACT

This section first presents an economic profile of the primary copper industry in general, and primary copper smelters in particular (Section 7.1). The data presented in the economic profile is then used in an economic analysis of the industry (Sections 7.2 and 7.3). The economic profile focuses on several primary copper smelter industry characteristics, such as: number and location of smelters, copper supplies, copper demand, competition, substitutes, and prices.

## 7.1 INDUSTRY ECONOMIC PROFILE

### 7.1.1 Introduction

Copper's utility stems from its qualities of electrical and thermal conductivity, durability, corrosion resistance, low melting point, strength, malleability, and ductility. Principal uses are in transportation, machinery, electronics, and construction.

The Standard Industrial Classification Code (SIC) definition of the primary copper industry is the processes of mining, milling, smelting, and refining copper. The primary copper smelters are included in SIC 3331 (Primary Smelting and Refining of Copper). Copper-bearing ore deposits and substantial amounts of copper scrap provide the raw materials for these processes.

In addition to producing copper, the industry markets by-product minerals and metals that are extracted from the ore deposits, such as silver, gold, zinc, lead, molybdenum, selenium, arsenic, cadmium, titanium, and tellurium. Many of the companies that own primary copper facilities also fabricate copper. Many of these same companies are also highly diversified and produce other metals, minerals, and fuels.

The standard under consideration directly affects only one of the four primary copper processes, namely smelting. However, the other three related processes are an integral part of the ownership and economic structures of copper smelters and therefore must be considered in determining industry impact. Mining and milling processes supplying a smelter will be secondarily affected by a smelter impact because transportation costs to an alternate smelter will add a sizeable business cost. Transportation costs for concentrate are significant because only 25 to 35 percent of the concentrate is copper and the remaining 75 to 65 percent that is also being transported is waste material. The same interdependence between smelter and refinery is not as critical because the copper content after leaving the smelter is typically 98 percent.

Even if there were no business dependencies among the processes, the available financial data for smelters are aggregated in consolidated financial statements which makes smelter data difficult to isolate. Thus, an economic analysis of copper smelters must be cognizant of the economic connection backward to the mines and forward through the refining stage.

### 7.1.2 Market Concentration

Fifteen pyrometallugical copper smelters exist in the United States. Copper is also produced in limited amounts by various hydrometallurgical methods which by-pass the smelting stage. These hydrometallurgical facilities are not being considered in the standard setting process. The 15 copper smelters have a capacity* of 1,722,600 megagrams** of copper. The hydrometallugical processes have a capacity of roughly 10 percent of the copper smelters' capacity.

Table 7-1 shows that the vast majority (approximately 89 percent) of smelting capacity is located in the southwestern States of Utah, Nevada, New Mexico, Arizona, and Texas, close to copper mines. The location is largely dictated by the need to minimize shipping distances of concentrates, which are normally 25 percent to 35 percent copper.

The 15 U.S. copper smelters are owned by seven large companies. All seven companies are integrated in that, to various degrees, they own some mining and milling facilities which produce copper concentrates for the smelters. Several smelters, apart from the concentrates from their own mines, buy additional concentrates from other mining and milling producers,

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*Capacity is not a static measure of a smelter since capacity can vary, for example, according to the grade of copper concentrates processed.

**1 megagram = 1.1 short tons.

Table 7-1. SMELTER OWNERSHIP, PRODUCTION, AND SOURCE MATERIAL ARRANGEMENTS[a]

| Smelter Name and Location | Ownership | 1980 Rated[1] Capacity (Mg) | 1979 Production (Mg) | 1980 Production (Mg) | Material Arrangements | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | 1979 | 1980 |
| Tacoma, WA | ASARCO, Inc. | 91,000 | 61,000[2] | 42,700[2] | Integrated | - | 31% | 20% |
| Hayden, AZ | | 182,000 | 96,000 | 59,000 | Custom | - | 25% | 44% |
| El Paso, TX | | 91,000 | 85,000 | 47,300 | Toll | - | 44% | 36% |
| Copperhill, TN | Cities Service Co. | 13,600 | 12,900[3] | 10,000[3] | Integrated[b] | - | 100% | 100% |
| White Pine, MI | Copper Range Co. Subsidiary of The Louisiana Land Exploration Co. | 52,000 | 39,770[4] | 32,500[4] | Integrated | - | 100% | 100% |
| Miami, AZ | Inspiration Consolidated Copper Company | 136,000 | 124,070[5] | 107,400[5] | Integrated | - | 35% | 58% |
| | | | | | Toll | - | 65% | 42% |
| McGill, NV | Kennecott Corp. | 45,000 | | | Integrated | - | 100% | 90% |
| Garfield, UT | | 254,000 | 296,000[6] | 259,300[6] | | | | |
| Hayden, AZ | | 71,000 | | | | | | |
| Hurley, NM | | 73,000[c] | 56,360 | 46,100 | Toll | - | | 10% |
| Magma (San Manuel, AZ) | Newmont Mining | 181,000 | 145,900[7] | 98,000[7] | Integrated | - | 100% | 100% |
| | | | | | Toll | - | | |
| Douglas, AZ | Phelps Dodge Corp. | 115,000 | | | Integrated | - | - | 75% |
| Ajo, AZ | | 64,000 | 283,000[8] | 324,100[8] | Custom | - | - | |
| Morenci, AZ | | 191,000 | | | Toll | - | - | 25% |
| Hidalgo, NM | | 163,000 | 91,000 | | | | | |
| Total Production | | 1,722,600[d] | 1,291,000 | 1,026,420 | Integrated | - | | 73% |
| Operating Rate | | | (74.9%) | (59.6%) | Custom | - | | 6% |
| | | | | | Toll | - | | 21% |

[a] Information primarily from corporate 10-K reports to the Securities and Exchange Commission.

[b] Estimate based on total copper sales for Cities Service minus the sales of its Arizona mines.

[c] Estimated to expand to 110,000 tons.

[d] Rated capacity excluding Anaconda smelter which was closed in 1980 (rated at 180,000 Mg).

smelt and refine the copper, and then sell it. This practice is referred to as custom smelting. Other smelters process (smelt and refine) the concentrates, and return the blister copper to mine owners for them to sell, a practice referred to as tolling. Some smelters perform both toll and custom smelting.

It is general industry practice for companies to operate their smelters as service centers at low profit margins to the owned mines. This acts to shift profits of an integrated operator to the mines, where depletion allowances exist. This maximizes profit to the overall operation. An implication of this policy is that the impact on profits from swings in copper prices is frequently manifest at the mines more than the smelters.

Table 7-1 lists the smelters, their corporate owners, capacities, 1979 and 1980 production amounts, and the distribution of integrated, custom, and toll smelting. Total production figures and the corresponding operating rates shown in Table 7-1 are compiled from corporate reports. Figures in Table 7-1 are adjusted to exclude capacity and production for the Anaconda smelter, which was closed in 1980. For 1979, the table shows a 74.9 percent operating rate. For 1980, the table shows that the industry operated at 59.6 percent of capacity. Production was down for 1980 due to an industry strike. Following the strike in 1980, production improved in 1981 to 1,380 gigagrams, for a capacity utilization rate of 80 percent.[9] Preliminary figures for 1982 from the Bureau of Mines show a decline in primary copper smelter production to 1,020 gigagrams, for a capacity utilization rate of about 59 percent.[10]

The three largest companies account for 78 percent of the entire smelting capacity. Phelps Dodge Corporation has the largest smelting capacity, followed by Kennecott Corporation and then ASARCO. The remaining four companies each have one smelter and in order of size are Magma (Newmont), Inspiration, Copper Range, and Copperhill (Cities Service).

The table also shows that 73 percent of total 1980 smelter production was from concentrate from integrated arrangements. Of the remaining concentrate, 21 percent was smelted on a toll basis and 6 percent smelted on a custom basis. Three of the eight companies process only their own copper concentrates.

### 7.1.3 Total Supply

Copper resources are defined as deposits which can be profitably extracted at a given price. Various estimates of U.S. copper resources show amounts ranging from 61.8 teragrams to 99.1 teragrams.* The capability of copper resources to meet future demand is dependent upon several factors; a principal one being the rate of growth in demand. The U.S. Bureau of Mines estimates that copper demand will grow at an annual growth rate of 3.0 percent to the year 2000 and that 30 percent of the demand will be supplied by scrap. Therefore, the likely primary copper demand over this period would be 55 teragrams compared with 92 teragrams of resources.[11] Consequently, U.S. supply appears adequate to the year 2000. Beyond the year 2000, demand is expected to strain supply sources. However, increased uses of old scrap and possible exploitation of sea nodules can supplement on-shore mining. In addition, microminiaturization, copper cladding, and other conservation methods will be more widely used to extend the supply of copper.

7.1.3.1 Domestic Supply. Primary refined copper output alone does not depict the entire supply of copper that is available for consumption in the United States. A large portion of copper scrap does not need to be resmelted or re-refined and is readily available for consumption. Copper is a durable material and, if it is unalloyed or unpainted, etc., it can be reused readily. Otherwise, it is resmelted or re-refined as described earlier. The ready availability of copper scrap as a secondary source of supply tends to be a stabilizing influence on producers' copper prices.

The total supply of copper available for consumption in any one year is therefore comprised of refined U.S. production, scrap not re-refined, net imports, and any changes in inventory of primary refined production from one year to the next.

The refined copper production in 1981 comprised 70.4 percent of total copper consumed in the United States; scrap not re-refined accounted for 32.0 percent and net refined imports 10.6 percent (total exceeds 100 percent due to stock changes).[12] Between 1970 and 1981, 67 percent of U.S. copper demand, excluding stock changes, was met from domestic mine production; 21

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*Teragram is 1.1 million short tons.

percent was from old scrap, and 12 percent from net imports. During these years, total U.S. demand for copper averaged 2,012,000 megagrams per year. Of this amount, 1,337,000 megagrams was from domestic production, 427,000 megagrams from scrap, and 248,000 megagrams from net imports.

Another statistic for describing the importance of scrap is to total the three stages (smelting, refining from scrap, and reuse of scrap) at which scrap can enter the production process, and compare the figures to total copper consumption. In 1981 the percentage of total consumed copper from scrap was 47.7, roughly the same as in recent years.

The 1981 refined copper production level was 1,956,400 megagrams. Although the average for the past several years has shown some improvement, total refined copper production has not returned to the 1973 peak level.

7.1.3.2 World Copper. According to the Bureau of Mines, the world reserve of copper in ore is estimated at 494,000 gigagrams of copper. In addition, an estimated 1,333,000 gigagrams of copper are contained in other land-based resources, and another 689,000 gigagrams in seabed nodules. The United States accounts for 19 percent of known copper reserves and 26 percent of other land-based copper resources.[13]

The United States is the leading copper producing and consuming country. Other major copper mining countries include: Chile, the U.S.S.R., Canada, Zambia, Zaire, Peru, and Poland. Although its copper mining activity is quite small, Japan is among the three largest countries in terms of copper smelting and refining. In 1981 the U.S. produced 18.8 percent of the world's mine production of copper, 16.5 percent of the smelter production, and 22.2 percent of the refinery production. The consumption of the world's refined copper by the U.S. amounted to about 21 percent. Table 7-2 shows U.S. production, world production, and the U.S. percent of world production for the years 1963 through 1981. Although the U.S. is essentially maintaining its consumption and production levels, world consumption and production is increasing. As a result, the U.S. share of world consumption and production shows a relative decrease.

In 1981 world consumption of refined copper rose 9 percent to 9,440 gigagrams.[14] Stocks of refined copper in the market economy countries increased 5 percent to 1,100 gigagrams.[15]

## Table 7-2. UNITED STATES AND WORLD COMPARATIVE TRENDS IN COPPER PRODUCTION: 1963-1981[16,17]

(Gigagrams)

| Years | Mine Production of Copper (Copper Content) | | | Smelter Production of Copper | | | Production of Refined Copper | | |
|---|---|---|---|---|---|---|---|---|---|
| | U.S. | World | U.S. as Percent of World | U.S. | World | U.S. as Percent of World | U.S. | World | U.S. as Percent of World |
| 1963 | 1100.6 | 4624.3 | 23.8 | 1176.3 | 4634.8 | 25.4 | 1709.5 | 5399.7 | 31.7 |
| 1964 | 1131.1 | 4798.6 | 23.6 | 1214.2 | 4851.4 | 25.0 | 1805.7 | 5739.0 | 31.5 |
| 1965 | 1226.3 | 4962.7 | 24.7 | 1300.9 | 5024.4 | 25.9 | 1942.1 | 6058.5 | 32.1 |
| 1966 | 1296.5 | 5215.9 | 24.9 | 1330.3 | 5167.0 | 25.7 | 1980.7 | 6322.2 | 31.3 |
| 1967 | 865.5 | 5057.6 | 17.1 | 782.3 | 4891.0 | 16.0 | 1384.9 | 6000.5 | 23.1 |
| 1968 | 1092.8 | 5456.5 | 20.0 | 1148.9 | 5507.8 | 20.9 | 1668.3 | 6658.6 | 25.1 |
| 1969 | 1401.2 | 5951.2 | 23.5 | 1438.3 | 5972.9 | 24.1 | 2009.3 | 7199.8 | 27.9 |
| 1970 | 1560.0 | 6387.3 | 24.4 | 1489.0 | 6309.5 | 23.6 | 2034.5 | 7577.8 | 26.8 |
| 1971 | 1380.9 | 6473.9 | 21.3 | 1360.8 | 6380.0 | 21.3 | 1780.3 | 7377.8 | 24.1 |
| 1972 | 1510.3 | 7071.5 | 21.4 | 1533.5 | 7003.2 | 21.9 | 2048.9 | 8068.0 | 25.4 |
| 1973 | 1558.5 | 7591.4 | 20.5 | 1582.1 | 7445.5 | 21.2 | 2098.0 | 8497.3 | 24.7 |
| 1974 | 1445.7 | 7885.6 | 18.3 | 1424.2 | 7933.6 | 18.4 | 1938.3 | 8851.5 | 21.9 |
| 1975 | 1282.2 | 6968.2 | 18.4 | 1357.5 | 7535.4 | 18.0 | 1610.7 | 8402.0 | 19.2 |
| 1976 | 1456.6 | 7525.3 | 19.4 | 1438.5 | 7839.6 | 18.3 | 1736.7 | 8322.3 | 20.9 |
| 1977 | 1364.4 | 7755.8 | 17.6 | 1346.8 | 8136.9 | 16.6 | 1706.9 | 8649.8 | 19.7 |
| 1978 | 1357.6 | 7618.3 | 17.8 | 1343.0 | 8017.5 | 16.8 | 1869.2 | 8791.9 | 21.3 |
| 1979 | 1443.6 | 7674.4 | 18.8 | 1396.0 | 8045.6 | 17.4 | 2013.8 | 8903.1 | 22.6 |
| 1980 | 1181.1 | 7656.3 | 15.4 | 1053.3 | 7938.9 | 13.3 | 1726.0 | 8971.0 | 19.2 |
| 1981 | 1538.2 | 8171.1 | 18.8 | 1377.7 | 8324.7 | 16.5 | 2037.6 | 9184.4 | 22.2 |
| Average Annual Compound Growth Rate (Percent) | | | | | | | | | |
| 1963-1973 | 3.54 | 5.08 | - | 3.01 | 4.85 | - | 2.07 | 4.64 | - |
| 1964-1974 | 2.48 | 5.09 | - | 1.61 | 4.77 | - | 0.71 | 4.43 | - |
| 1971-1981 | 1.09 | 2.36 | - | 0.12 | 1.03 | - | 1.36 | 2.21 | - |

Note: One gigagram = 1,000 megagrams. One megagram (1,000 kilograms) equals 1.102311 short tons (907.185 kilograms = 2000 pounds avoirdupois, where one pound avoirdupois equals 0.453592 kilogram or 453.5924 grams).

### 7.1.4 U.S. Total Consumption Of Copper

Total copper consumed in the United States over the last 12 years has fluctuated considerably but shows an overall upward trend. However, copper consumption has not returned to its 1973 peak. This conclusion is derived from data on copper consumption from refineries and copper consumption from refineries plus scrap.

Table 7-3 shows each set of data for the years 1970 through 1981. The 5-year averages in gigagrams for copper consumption from refineries has increased by 6.9 percent (1972 through 1976 is 1,891.9 and 1977 through 1981 is 2,021.5 ). Five-year scrap consumption has shown an increase of 5.1 percent, from 848.6 gigagrams for the 1972 to 1976 period, to 892.3 gigagrams for the 1977 to 1981 period. There are signs that the consumption of scrap has begun to increase over the last few years.

The Bureau of Mines forecasts a long-range overall consumption growth rate to the year 2000 of 3.0 percent per year. The combined 3.0 percent growth rate is composed of a 2.4 percent growth rate for primary copper, and a 4.8 percent growth rate for secondary copper.[19]

7.1.4.1 Demand By End-Use. Refined copper and copper scrap are further processed in a number of intermediate operations before the copper is consumed in a final product. Refined copper usually consists of one of the following shapes: cathodes, wire bars, ingots, ingot bars, cakes, slabs, and billets. These shapes plus the copper scrap then go to brass mills, wire mills, foundries, or powder plants for subsequent processing. The copper is frequently alloyed and transformed into other shapes such as sheet, tube, pipe, wire, powder, and cast shapes. Ultimately, the copper is consumed in such shapes in five market or end-use categories. The Copper Development Association, Inc. uses the following categories: building construction, transportation, consumer and general products, industrial machinery and equipment, and electrical and electronic products.

Table 7-4 shows the demand for copper in each of these five markets over the 12-year period 1970 through 1981. The total figures for these five markets will not equal the total consumption figures of Table 7-3

## Table 7-3. U.S. COPPER CONSUMPTION[18]

(Gigagrams)[a]

| | 1970 | 1971 | 1972 | 1973 | 1974 | 1975 | 1976 | 1977 | 1978 | 1979 | 1980 | 1981[b] |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Consumption of Refined Copper | 1859.8 | 1833.0 | 2029.5 | 2220.6 | 1998.6 | 1398.9 | 1811.7 | 1989.2 | 2196.5 | 2164.1 | 1876.3 | 1890.6 |
| Consumption of Scrap[c] | 811.6 | 859.5 | 946.0 | 956.4 | 878.0 | 662.4 | 800.4 | 847.6 | 881.1 | 984.3 | 860.0 | 888.6 |
| Total Consumed Copper | 2671.4 | 2692.5 | 2975.5 | 3177.0 | 2876.6 | 2061.2 | 2612.2 | 2836.8 | 3077.5 | 3148.4 | 2727.3 | 2779.1 |
| % of Total as Scrap | 30.4 | 31.9 | 31.8 | 30.1 | 30.5 | 32.1 | 30.6 | 29.9 | 28.6 | 31.3 | 31.5 | 32.0 |
| 1972-1976 and 1977-1981 averages for consumption refined | | | 1891.9 | | | | | 2021.5 | | | | |
| 1972-1976 and 1977-1981 averages for consumption of scrap | | | 848.6 | | | | | 892.3 | | | | |
| 1972-1976 and 1977-1981 averages for total copper consumption | | | 2740.5 | | | | | 2913.8 | | | | |

[a] 1 gigagram = 2.2 million pounds.

[b] Preliminary.

[c] Without having to be refined again.

## Table 7-4. U.S. COPPER DEMAND BY MARKET END USES[20]

(Gigagrams)[a]

| Market | 1970 | 1971 | 1972 | 1973 | 1974 | 1975 | 1976 | 1977 | 1978 | 1979 | 1980 | 1981[b] |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Building Constr. | 748 | 830 | 923 | 998 | 795 | 634 | 780 | 888 | 956 | 951 | 774 | 816 |
| Transportation | 267 | 307 | 345 | 402 | 332 | 265 | 372 | 412 | 422 | 388 | 270 | 310 |
| Consumer and General Products | 369 | 358 | 417 | 413 | 402 | 316 | 328 | 340 | 388 | 420 | 342 | 376 |
| Industrial Machinery and Equipment | 560 | 540 | 601 | 640 | 563 | 404 | 483 | 518 | 551 | 565 | 501 | 537 |
| Electrical and Electronic Products | 717 | 742 | 773 | 838 | 766 | 541 | 683 | 767 | 801 | 853 | 754 | 774 |
| Total | 2661 | 2777 | 3059 | 3291 | 2858 | 2161 | 2646 | 2925 | 3118 | 3177 | 2641 | 2813 |
| 5-Year Average Demand[c] | | | | | | | | | | | | |
| Building Construction | | | 826.0 | | | | | 877.0 | | | | |
| Transportation | | | 343.4 | | | | | 360.4 | | | | |
| Consumer and General Products | | | 375.2 | | | | | 373.2 | | | | |
| Industrial Machinery and Equipment | | | 538.2 | | | | | 534.4 | | | | |
| Electrical and Electronic Products | | | 720.2 | | | | | 789.8 | | | | |

[a] 1 Gigagram = 2.2 million pounds.

[b] Preliminary.

[c] (1972-1976, and 1977-1981).

due to the effects of stock changes and imports on fully fabricated copper products.

A look at the 5-year average demand shows that there has been an increase in three out of the five markets. The building industry market sales showed a gain of 6.2 percent. The transportation market shows a gain of 5.0 percent. An increase of 9.7 percent occurred in the electrical and electronic product markets. The demand for electrical equipment has risen because of increased emphasis on safety, comfort, recreation, and a pollution-free environment. Automation, including the use in computers, has also boosted the use of copper.

Substitution of other materials, coupled with the recession, has caused the slight drop of less than 1 percent in the consumer and general products markets. The 1 percent decline in the industrial machinery and equipment market is largely due to the impact of the recession.

The Bureau of Mines estimates that the most growth in copper demand will occur in the electrical and electronic products industries, consumer and general products, and building construction. Copper is an important metal in electric vehicles. If electric vehicles become popular, this would be a source of increased demand for copper. General Motors plans to produce an electric family car for mass marketing in the mid-1980's. A conventional internal combustion automobile contains from 6.8 to 20.4 kg of refined copper, whereas electric vehicles use much more copper. The Copper Development Association estimates range from 45.4 kg to 90.7 kg, with an average nearer to 45.4 kg.[21]

Another potential area for growth is in the solar energy industry. Presently, the extent of this sector is relatively modest, consuming approximately 4,500 Mg/yr of copper in the U.S. However, consumption in this sector has the potential to climb considerably.

In addition, the U.S. military demand for copper is expected to increase. Increased military expenditures will have a significant impact on copper demand because copper is an important element in modern electronic weaponry. During heavy rearmament periods the military demand for the metal has reached 18 percent of copper mill shipments. Although military demand is not expected to return to the record high 18 percent level, analysts do expect a large increase in military requirements for copper from the low level in 1979 of less than 2 percent.[22]

The demand picture in the United States may receive a boost from the federal government. The government is committed to eventually acquire 1.1 gigagrams of copper for its currently depleted strategic stockpile. The previous stockpile was largely depleted in 1968; the final sale was in 1974 after copper prices had soared. Further Congressional action is necessary to implement and fund the purchase plan.

7.1.4.2 Substitutes. Substitutes for copper are readily available for most of copper's end uses. Copper's most competitive substitute is aluminum. Other competitive materials are stainless steel, zinc, and plastics. Aluminum, because of its high electrical conductivity, is used extensively as a copper substitute in high voltage electrical transmission wires. Aluminum has not been used as extensively in residential wiring because of use problems, and minimal savings.

Aluminum is also potentially a substitute for copper in many heat exchange applications. For example, automobile companies are still experimenting with the use of aluminum versus copper in car radiators. When copper prices are high, or copper supply is limited, cast iron and plastics are used in building construction as a copper pipe substitute. A relatively new substitute for copper is glass, which is used in fiber optics in the field of telecommunications.

### 7.1.5 Prices

Numerous factors influence the copper market, and thus the price of refined copper. These factors include: production costs, long-run return on investment, demand, scrap availability, imports, substitute materials, inventory levels, the difference between metal exchange prices and the refined price, and federal government actions (e.g., General Services Administration stockpiling and domestic price controls).

Among the many published copper price quotations, two key price levels are: 1) those quoted by the primary domestic copper producers and 2) those on the London Metal Exchange and reported in Metals Week, Metal Bulletin, and the Engineering and Mining Journal. The producers' price listed most often is for refined copper wirebar, f.o.b. refinery. The London Metal Exchange price, referred to as LME, is also for copper sold as wirebar. The LME is generally considered a marginal price reflective of short-term supply-demand conditions, while the producer price is more long-term and stable and often lags the LME price movement.

Copper is also traded on the New York Commodity Exchange (Comex). Arbitrage keeps the LME price and the Comex price close together (with minor price differences due to different contract terms on the two exchanges, and a transportation differential).

Table 7-5 shows the LME, the U.S producer price, and the U.S. producer price adjusted to a 1982 constant price for the years 1970 through 1982. Data were obtained from U.S. Bureau of Mines publications.

Several points can be observed from the table with respect to the LME price versus the U.S. producer price: (1) the LME price has had wider swings than the producer price; (2) in the past when both prices are relatively high, the LME price has been considerably higher than the producer price, while during relatively low price periods, the producer price has been moderately higher than the LME price; and (3) in recent years a marked change appears to be taking place away from a two-price system and toward a one-price system, with the difference between the LME and the U.S. producer price accounted for only by a transportation differential. These earlier situations had reoccurred repeatedly over the past 20 years. One other point about the table should be mentioned, although unrelated to the relationship of the LME to the producer price. The producer price has not kept pace with general inflation.

In theory, the U.S. producer price should be somewhat higher than the LME price since ocean transport costs must be incurred to get the refined copper to the U.S. However, this relationship appears to hold only during slack price periods. When LME prices are high, the producers do not raise their prices as much, which in theory appears contrary to profit maximization. Explanations offered for such behavior include: the producers' fear of long-run substitution for copper if the producers raised the price to the fabricators, high profits for integrated fabricators while reducing supply to nonintegrated fabricators, past fears of government stockpile sales that would reduce prices, and fear of the return of government intervention through price controls.

## Table 7-5. AVERAGE ANNUAL COPPER PRICES[23,24,25]
## (cents per kg)[a]

| Year | LME[b] | U.S Producer Price[c] | U.S. Producer Price 1982 Constant Price[d] |
|---|---|---|---|
| 1970 | 138.6 | 128.0 | 290.9 |
| 1971 | 106.7 | 114.4 | 248.7 |
| 1972 | 106.7 | 112.6 | 234.6 |
| 1973 | 178.0 | 130.9 | 256.7 |
| 1974 | 204.8 | 170.1 | 303.8 |
| 1975 | 123.4 | 141.2 | 231.5 |
| 1976 | 140.6 | 153.1 | 239.2 |
| 1977 | 130.7 | 147.0 | 216.2 |
| 1978 | 136.2 | 146.3 | 200.4 |
| 1979 | 198.2 | 205.3 | 259.9 |
| 1980 | 218.5 | 225.3 | 262.0 |
| 1981 | 174.7 | 187.2 | 199.1 |
| 1982[e] | 147.4 | 162.8 | 162.8 |

[a] To convert from cents/kg to cents/lb, multiply by 0.454.

[b] London Metal Exchange "high-grade" contract.

[c] U.S producer price, electrolytic wirebar copper, delivered U.S destinations basis.

[d] Adjusted to 1982 constant price by applying implicit price deflator for gross national product (1972 = 100).

[e] Preliminary.

The cost of producing copper is one of the elements that influences the price of copper. Considerable data exist to validate the point that the long-run economic cost of producing copper is increasing.[26] During the early 1970's the capital costs per megagram of annual capacity for developing copper from the mine through refining stage were $2,000 to $2,500, and by the late 1970's had risen sharply to $7,200 to $7,700. Estimates are that a price of $2.76 per kg to $3.30 per kg for refined copper would be needed to support such new capital outlays.

The above costs are for conventional pyrometallurgical smelting. The newer smelting processes such as Noranda and Mitsubishi offer some capital cost savings at that stage due to lower pollution control costs. The hydrometallurgical processes also require less capital. However, the mining costs are the highest part of overall development costs for which limited cost saving techniques exist. The mine development costs in the U.S. have risen significantly, largely as a result of the shifting from higher to lower grades of available copper ores and sometimes remote locations that require infrastructure costs for towns, roads, etc.

In 1979, the Bureau of Mines analyzed 73 domestic copper properties to determine the quantity of copper available from each deposit and the copper price required to provide each operation with 0 and 15 percent rates of return. The Bureau estimates that a copper price of $4.56 per kg would be required if all properties, producing and nonproducing, were to at least break even. The average break-even copper price for properties producing in 1978, $1.46 per kg, was about equivalent to the average selling price for the year. At this price, analysts calculate that only 25 properties could either produce at break-even or receive an operating profit. Of these properties, only 12 could receive at least a 15 percent rate of return.

Annual domestic copper production, from 1969 to 1978, averaged 1,337,000 megagrams. According to this study, in order to produce at this level and receive at least a 15 percent rate of return, a copper price of $1.81 per kg is required. If the United States were to produce the additional 248,000 megagrams that were imported each year over this period, a copper price of $1.94 would be necessary.[27] The report concludes that increases in copper prices are required in order for many domestic deposits to continue to produce.

It has been suggested that long-term potential for higher prices, plus the high cost of new capacity are significant reasons for the increased purchases several years ago of copper properties by oil companies. The reasoning is that oil companies need places for heavy cash flows, and diversification to other products is desirable. The oil companies reportedly can wait for expected copper price increases to obtain their return. Further, by purchasing existing facilities, rather than building new capacity, they avoid the escalating new capacity costs. However, more recently, some oil companies seems to be rethinking their investments in copper.

As shown below, U.S. oil (and gas) companies own or have major interests in many of the largest domestic copper producers:

1. Amax - Approximately 20 percent owned by Standard Oil of California
2. Anaconda - Owned by Atlantic Richfield Company (ARCO)
3. Cities Service - Also a primary copper producer
4. Copper Range - Owned by Louisiana Land and Exploration Company
5. Cyprus Pima Mining Company - Standard Oil Company (Indiana)
6. Duval - Owned by Pennzoil Company
7. Kennecott - Standard Oil of Ohio (British Petroleum)

These copper producers own or control a large portion of domestic copper reserves, mine production, and U.S. refinery capacity. Their investment in the copper industry is significant, and thus they must expect higher prices and substantial profits in the future.

## 7.2 ECONOMIC ANALYSIS

### 7.2.1 Introduction

This section presents the economic impact analysis of the arsenic NESHAP for the high arsenic primary copper smelter. The 14 other primary copper smelters are classified as low arsenic smelters and are discussed in a separate analysis.

The principal economic impacts analyzed are: the ability of the smelter to increase copper prices in response to an increase in costs due to the arsenic standard, and the impact on profits if part or all of the costs cannot be passed on in the form of price increases. Section 7.2.3 presents the methodology, Section 7.2.4 presents the impact on prices, Section 7.2.5 presents the impact on profits, and Section 7.2.6 presents a discussion of capital availability.

### 7.2.2 Summary

In 1982, the copper producers experienced one of the worst years in recent history. Such a situation cannot be used as the foundation to examine the long-term economic impact of the potential arsenic NESHAP. Therefore, the economic analysis is based on a more normal condition for the industry.

If the smelter attempts to pass control costs forward in the form of a price increase, the price increases would be relatively modest and would range from 0.7 percent to 1.1 percent, depending upon the capacity utilization rate and the price that is used.

If control costs are absorbed and profit margins reduced, the profit reductions would range from a high of 10.7 percent to a low of 3.1 percent, depending upon the capacity utilization rate and the price that is used.

The capital costs of the control equipment are not minor amounts. However, ASARCO is a major publicly-held corporation with a good credit rating and good access to the capital markets. The incremental capital cost of the control equipment for the Tacoma smelter alone does not present a major obstacle. If capital costs were incurred at ASARCO's two other smelters, the capital would still be likely to be available.

### 7.2.3 Methodology

The purpose of this section is to explain in general terms the methodology used in the analysis. Each of the appropriate sub-sections explains the methodology in more detail. No single indicator is sufficient by itself to use for decision making purposes about the smelter. Therefore the methodology relies on several indicators which in total can be used to draw conclusions.

The methodology has three major parts. The first part is an analysis of price impacts. The analysis of price impacts introduces an upper limit on the problem and provides a benchmark to make evaluations on a relatively uncomplicated basis. A price increase is the "worst case" from the viewpoint of a consumer of copper. The second major part of the methodology is an analysis of profit impacts. The analysis of profit impacts introduces a lower limit on the problem and is the "worst case" from the viewpoint of the firm. The individual characteristics of the smelter increase in importance and are incorporated to a greater extent. The third and final part is an analysis of the availability of capital to purchase the control equipment.

Firms in the copper industry face a wide variety of variables that in the aggregate determine the economic viability of the firm generally and a smelter specifically. The variables can be grouped in four broad categories. The categories are described here separately and in a simplified manner for discussion purposes. However, there is a close interrelationship among the four categories and changes in one will have implications for the others. The four broad categories which encompass the variables that in turn determine the economic viability of the smelter are described below.

1) Macro-economic conditions. The two most prominent variables in this category are copper prices and copper demand. By-products and co-products represent a significant source of revenues for most copper operations. Therefore, in addition to the price of copper, the price of by-products and co-products also influence an assessment of economic viability. Common by-products and co-products of copper production include: gold, silver, molybdenum, and sulfuric acid. Other byproducts include selenium, tellurium, and antimony. A by-product that sets Tacoma apart from other smelters is arsenic (arsenic trioxide and metallic arsenic). For ease of presentation and in order to present a conservative analysis, by-products and co-products are not considered explicitly in the analysis. Another important variable, though somewhat less visible, is government actions such as tax policy,

stockpiling and price controls. The government variable includes the U.S. Government, as well as foreign governments. For example, consider that a report by the U.S. Bureau of Mines has stated that at least 40 percent of the total mine production of copper in market economy countries was produced by firms in which various foreign governments owned an equity interest.[28]

2) Environmental regulations. Since roughly 1970, environmental regulations have evolved to the point that they have become a major variable that must be considered in the corporate decision-making process. Here again, government actions are important.

3) Corporate organizational strategy. This category includes the corporation's strategy with respect to variables such as remaining or becoming an integrated copper producer versus a non-integrated copper producer, or in the extreme, leaving the industry entirely.

Many of the companies that produce refined copper are integrated producers; that is, they own the facilities to treat copper during each of the four principal stages of processing: mining, milling, smelting, and refining. Also, several of the producers are integrated one additional step into the fabrication of refined copper. However, not all companies in the copper industry are integrated producers. There are companies that only mine and mill copper ore to produce copper concentrate, and then have the copper concentrate smelted and refined on a custom basis (the smelter takes ownership of the copper) or on a toll basis (the smelter charges a service fee and returns the copper to the owner). Information presented earlier in Table 7-1 has shown that about 25 percent of the concentrate processed by ASARCO is from integrated mines owned by the company. However, the major portion, the remaining 75 percent, of the concentrate it processes is smelted and refined on a toll or custom basis. The existence of both integrated and nonintegrated production introduces a complex economic element into this analysis. That complex economic element manifests itself in the choice of the appropriate profit center. This standard affects only one stage of the production process (smelting) in a direct way, but has indirect effects on the other stages.

For accounting purposes, integrated producers frequently view the smelter as a cost center, rather than a profit center. However, in an economic sense the smelter provides a distinct contribution to the production process that ultimately allows a profit to be earned although that profit may be realized for accounting purposes at another stage, such as the mine or refinery.

4) Competition. Mines have long-run flexibility in deciding where they will send their copper concentrate for smelting. Therefore, copper smelters face competition from three sources: other existing domestic smelters, new smelters that may be built, and foreign smelters, especially Japanese. Other competition, though less direct, is also important. For example, scrap and substitutes present competition.

Japan is a major force among copper producing countries in terms of its volume of smelting, refining, and fabrication of copper. However, Japan does not have copper ore deposits of any noteworthy size. Therefore, it must import concentrates in order to supply its smelting, refining, and fabricating facilities. Japan seeks concentrates from many countries, including the United States. Japan's ability to be competitive with domestic smelters for U.S. concentrates is indicated by the contractual arrangements it has established with Anamax and Anaconda to purchase concentrates. Also, the Japanese smelters have approached many other copper mine owners in the United States. For example, Cyprus Corporation is reported to have seriously considered shipping concentrates from its Bagdad mine to Japan.

The cost to transport concentrates across the Pacific Ocean is significant. The fact that Japanese smelters can compete with U.S. smelters, in spite of the costs to transport concentrates across the Pacific Ocean, is quite noteworthy. One factor that explains the Japanese ability to compete is that Japanese smelters are newer than U.S. smelters and, in theory, should be more cost competitive. Other factors that operate to the advantage of Japanese smelters, including a tariff mechanism, are described later.

The existence of competition for concentrates introduces what is commonly referred to as a "trigger" price. The "trigger" price is that price which triggers or provides an economic incentive for the supplier of concentrate to change to another smelter and refinery. If a given smelter charges a service fee in excess of competing smelters, that smelter will lose business and eventually be forced to cease operations. In the case of new smelters or expansions, the new process facilities will not be built. Faced with an increase in costs, a smelter could respond using one of three options, or any combination of the three. First, the smelter could pass the costs forward in the form of a price increase. Two important considerations with respect to a price increase are: the prices of competitors in the copper business, and the elasticity of demand for the end-users of copper. For example, even if all copper producers experience the same increase in costs, at some point the end users of copper will consider changing to a substitute. Second, the smelter could absorb the cost increase by reducing its profit margins, thereby reducing its return on investment (ROI). If the smelter's profit margins are reduced significantly it will cease operation. Third, the smelter could pass the costs back to the mines by reducing the price it is willing to pay for concentrate. An important consideration in setting the service fee a smelter charges for custom or toll smelting is that the concentrate may be shipped elsewhere, such as to Japan. Market conditions suggest that the option of passing costs back to the mines does not seem feasible at this time, due to the existence of excess smelting capacity.

7.2.3.1 Japanese Tariff mechanism. One example of foreign government assistance to the copper industry occurs in Japan. Japanese copper producers operate under a system that permits the payment of a premium for concentrates, which is then recovered through a premium for refined copper due to a protected internal market supported by a high tariff. Japan imposes high import duties on refined, unwrought copper while allowing concentrates to be shipped into the country duty-free. Duty on refined unwrought copper in 1981 was 8.2 percent of the value of the copper, including freight and insurance, as opposed to a U.S. customs duty of 1.3 percent of the value of copper. The import duties allow Japanese producers to sell their refined copper in Japan at an artificially high price and still remain competitive with foreign producers.

Specifically, copper concentrates and ore imported into Japan are free of duty. Refined copper imported into Japan is subjected to a tariff of 15,000 yen/Mg.[29] Using a December 15, 1980, exchange rate of $0.004633/yen, the tariff was $0.0849/kg. Refined copper may be duty-free under the preferential tariff, subject to certain limitations.

As a result of the tariff situation, Japanese copper producers can pay a premium to attract concentrates and can recover the premium through a premium on the price of the refined copper used in Japan. If the refined copper is returned to the customer outside of Japan, the premium on the price of refined copper is not recovered because world prices would prevail in this case, rather than the protected internal Japanese producer price. As a result, the principal interest of the Japanese copper producers is in producing copper for internal consumption. Toll smelting in Japan is generally used as a means of balancing inventories. The absence of a tariff on ore and concentrates encourages companies to import ore into Japan. The presence of a tariff on refined copper and the costs of holding metal in Japan discourage companies from importing refined copper into Japan.

The Japanese tariff on refined copper, combined with the cost of holding the metal until users have a demand for it, provides an extra margin for domestic copper producers. The Japanese producers can charge what the market will bear for their copper and still remain competitive with the importers. The loss incurred by Japanese producers in charging toll customers low processing rates is covered by the extra margin of profit realized by charging prices for domestic refined copper at competitive import levels.

Robert H. Lesemann (industry expert, formerly with Metals Week, now with Commodities Research Unit), in an affidavit for the Federal Trade Commission, outlined the situation in September 1979:

> It is generally true that operating costs of U.S. smelters are the same as smelters in Japan, Korea, and Taiwan. The competitive advantage is without doubt due to the subsidies outlined above. Thus, while the terms of the Nippon-Amax deal have not been revealed, the treatment charge is likely well below the operating cost levels of U.S. smelters.[30]

7.2.3.2 Other Japanese advantages. The tariff mechanism described above is one example of government assistance to the Japanese copper industry. Another example is provided by the Japanese government's approval of a brass rod production cartel. In an effort to reduce stocks and boost profit margins for the ailing Japanese brass rod industry, the government approved the formation of a temporary cartel to cut production.[31]

Apart from government assistance, other reasons are cited for the advantage of the Japanese copper industry over the U.S. copper industry.

Additional reasons include:

- A high debt-to-equity ratio--a typical Japanese smelter may have a debt-to-equity ratio of 0.8 to 0.9.[32,33,34]
- Lower labor rates--Japanese hourly rates in the primary metals industry were estimated to be about two-thirds of the U.S. rate in 1978.[35]
- By-product credits--the market for by-products, sulfuric acid, and gypsum is better in Japan than in the United States and reduces operating costs significantly.[32]

### 7.2.4 Maximum Percent Price Increase

Insight into the economic impact of the arsenic NESHAP can be gained by examining the maximum percentage copper price increase that would occur if all control costs were passed forward in the form of a price increase. A complete pass forward of control costs may not be possible in every case, and later in the analysis this assumption is relaxed. However, the initial assumption that a complete pass forward is possible in every case introduces a common reference point, which then facilitates comparisons of various control alternatives and scenarios.

The maximum percentage price increase is calculated using a simplified approach, for ease of presentation, that divides annualized control costs by the appropriate production and further divides that result by the refined price of copper, with the result expressed as the necessary percentage price increase per kilogram. The above approach does not consider the investment tax credit, and thus is a conservative approach that will tend to overstate the impact of the control costs. Other approaches could be used to determine price increases. For example, a net present value (NPV) approach could be used. A net present value approach determines the revenue increases necessary to exactly offset the control costs, such that the NPV of the plant remains constant. An NPV analysis can also take into account the investment tax credit, depreciation over the applicable time period, income taxes, operating and maintenance costs, and the time value of money. Although the NPV approach is a more sophisticated calculation, the two approaches yield similar results. Therefore, the first method is preferable in this particular case due to its straightforward nature, ease of presentation, and reasonable results.

Table 7-6 first shows the cost increase, and then the maximum percentage price increase, of arsenic controls for the ASARCO-Tacoma primary copper smelter. The increase in the cost of production is shown for two capacity utilization rates, 100 percent and 80 percent. The advantage of presenting two capacity utilization rates is in the conduct of sensitivity analysis. A rate of 100 percent is optimistic, but is useful here as a reference point. A rate of 80 percent is more likely and as noted in Section 7.1 this is the approximate industry average utilization rate achieved in 1981. As described earlier, capacity for Tacoma is presented as 91,000 Mg per year. An 80 percent capacity utilization rate represents production of 72,800 Mg, which is roughly the amount of production achieved at Tacoma during 1981. For 1982, the industry average capacity utilization rate was substantially lower at 59 percent. However, no analysis is shown here of the impact of control costs at a 59 or 60 percent utilization rate because regardless of control costs, a rate of 60 percent is damaging even as a baseline condition. For example, based on a capacity at Tacoma of 91,000 Mg per year, for the years from 1975 to 1981 only the strike-year of 1980 had a capacity utilization rate of 60 percent or below. The purpose of showing the increase in production cost is to supplement the maximum percentage price increase. One advantage of reviewing the cost increase is that it is dependent only on the capacity utilization rate, and is not affected by the refined price of copper. A second advantage is that it is not affected by the choice of the profit center. Two points should be observed from the cost increases:

1) Although any cost increase is undesirable from the firm's viewpoint, the amount of the cost increase is relatively modest. Although the cost increase is relatively modest several factors should be mentioned. First, copper is a commodity, which means that product differentiation is not possible and thus competition is based almost exclusively on price. The copper producers can be characterized as price-takers and thus no individual producer controls the marketplace. Therefore, in an industry that competes based on price, the cost of production becomes exceptionally important. Second, copper is traded on an international basis and thus domestic producers compete among themselves, as well as against foreign producers that may not experience the same cost increases. Finally, copper is faced with a significant threat from substitutes such as aluminum and plastic.

Table 7-6. COST INCREASE AND MAXIMUM PERCENT PRICE INCREASE OF ARSENIC CONTROLS FOR THE HIGH-ARSENIC PRIMARY COPPER SMELTER

|  |  | Cost Increase (¢/kg) |  | Maximum Percent Price Increase |  |  |  |
|---|---|---|---|---|---|---|---|
|  |  |  |  | 187 ¢/kg |  | 220 ¢/kg |  |
| Smelter | Annual Capacity (Mg) | 100% Capacity Alternative II | 80% Capacity Alternative II | 100% Capacity II | 80% Capacity II | 100% Capacity II | 80% Capacity II |
| ASARCO-Tacoma | 91,000 | 1.6 | 2.0 | 0.9 | 1.1 | 0.7 | 0.9 |

2) The difference between the two capacity utilization rates, in terms of the size of the cost increase, is minimal.

Table 7-6 also shows maximum percentage price increases. The purpose of reporting the maximum percentage price increase figures is to add perspective to the cost increase figures. Results are shown for two refined copper prices (187 cents per kg and 220 cents per kg), and for the same two capacity utilization rates presented earlier, 100 percent and 80 percent. The price increase assumes the firm is an integrated producer. The average annual price for refined copper over the past 5 years, from 1978 to 1982, has been approximately 187 cents per kg. The price of copper is difficult to predict, and therefore prudence suggests examination of a second price. As shown previously in Section 7.1, the highest average annual current dollar price for refined copper was 225.3 cents per kilogram, achieved in 1980. (The year 1980 was marked by an industry strike and reduced production.) Therefore, 220 ¢/kg is used to represent a price that, based on the results of past years, appears optimistic. An alternative "pessimistic" price is not presented because even the baseline results are highly likely to be damaging, and thus the addition of control costs would merely reinforce an obvious conclusion. A ready example of the impact of a price significantly below 187¢/kg was provided in 1982 when the average price was about 163 ¢/kg and large segments of the industry closed for sustained periods. The analysis of the results for the maximum percentage price increase figures is similar to the analysis discussed above for the cost increase figures. The results show that the size of the maximum percentage price increase is modest, and there is minimal variation. At a price of 187¢/kg, the price increase is 0.9 percent at a 100 percent capacity utilization rate, and 1.1 percent at an 80 percent capacity utilization rate. At a price of 220¢/kg, the price increases are 0.7 and 0.9 percent, respectively. However, as mentioned above, although the price increases are modest, two constraining influences are foreign competition and substitutes.

### 7.2.5 Profit Impacts

Apart from the calculation of maximum percentage price increase, additional insight into the economic impact of the arsenic NESHAP can be gained by making the opposite assumption from maximum percent price increase;

that is, zero percent price increase, or complete cost absorption. The assumption of complete control cost absorption provides a measure of the reduction in profits if the control costs are absorbed completely.

Assuming control costs are absorbed, the critical element in an analysis of profit impacts is the profit margin. The larger a firm's profit margin, the greater is the firm's ability to absorb control costs and earn an acceptable rate of return on investment (ROI), and thus continue operation. The profit margin is simply the difference between price and cost. As mentioned in an earlier section, the central issue becomes the choice of an appropriate profit center and its corresponding price and cost. The processing of virgin ore into refined copper involves four distinct steps: mining, milling, smelting, and refining. Although the four steps are often joined to form an integrated business unit, they are not inextricably bound together in an economic sense. For example, it is not uncommon for mines to have their concentrate toll smelted and refined. The difficulty that this variability presents in terms of an assessment of the impact of the arsenic standard is in the method of assigning the costs.

This report presents an analysis of profit impacts using two methods. The first method assumes the smelter is fully integrated. The objective of this method is to permit a ready, though simplified, examination of profit impacts. With the first method as a foundation, the second method introduces more smelter specific variables into the analysis in an effort to focus more sharply on the complex organizational structure of the industry.

7.2.5.1 Method One. As mentioned above, the critical element in an examination of profit impact is the profit margin. Therefore, an examination of profit margins for members of the industry is necessary, and accordingly is presented below. Table 7-7 shows the revenues and operating profit (before tax) for each of the seven producers for the 5-year period from 1977 to 1981. Table 7-7 also shows the percentage return on sales, which is operating profit divided by revenues. The revenue and operating profit figures are for the business segment within the company that includes copper. The use of business segment information provides a closer representation of the results for copper than would the use of the consolidated results for the company. The reason for this is that for several of the

Table 7-7. BUSINESS SEGMENT RETURN ON SALES FOR COPPER COMPANIES[a]

($ $10^3$)

| | Year | ASARCO | Cities Service | Copper Range[b] | Inspiration | Kennecott | Magma[c] | Phelps Dodge |
|---|---|---|---|---|---|---|---|---|
| Revenues | 1977 | 733,293 | 184,000 | NA | 95,676 | NA | NA | 453,184 |
| | 1978 | 849,002 | 241,500 | 64,600 | 101,251 | 683,000 | 274,137 | 446,970 |
| | 1979 | 1,339,917 | 276,300 | 89,300 | 136,849 | 1,091,400 | 381,512 | 618,188 |
| | 1980 | 1,440,220 | 224,100 | 83,900 | 178,004 | 987,400 | 287,581 | 714,591 |
| | 1981 | 1,153,022 | NA | NA | NA | NA | 328,842 | 706,404 |
| Operating Profit[d] | 1977 | 65,919 | (38,600) | NA | (9,994) | NA | NA | 52,831 |
| | 1978 | 112,474 | (23,900) | (6,600) | (6,235) | (100) | 13,601 | 63,738 |
| | 1979 | 225,763 | 25,400 | 10,000 | 9,889 | 164,000 | 67,252 | 159,428 |
| | 1980 | 145,286 | 16,300 | 1,800 | (6,563) | 131,400 | 11,522 | 95,439 |
| | 1981 | 68,364 | NA | NA | NA | NA | (15,658) | 27,618 |
| Profit/ Revenues (percent) | 1977 | 9.0 | (21.0) | NA | (10.4) | NA | NA | 11.7 |
| | 1978 | 13.2 | (9.9) | (10.2) | (6.2) | 0 | 5.0 | 14.3 |
| | 1979 | 16.8 | 9.2 | 11.2 | 7.2 | 15.0 | 17.6 | 25.8 |
| | 1980 | 10.1 | 7.3 | 2.1 | (3.7) | 13.3 | 4.0 | 13.4 |
| | 1981 | 5.9 | NA | NA | NA | NA | (4.8) | 3.9 |
| | Average | 11.0 | (3.6) | 1.0 | (3.3) | 9.4 | 5.5[e] | 13.8[f] |

[a] Business segments contain other products in addition to copper.

[b] The Louisiana Land and Exploration Company.

[c] Profit is net income after tax in this case.

[d] Before interest and tax.

[e] Would yield 7.9 percent if adjusted to before tax with an effective tax rate of 30 percent.

[f] Imputed profit on intersegment sales for 1977 to 1981 would yield average return of 17.5 percent.

firms copper represents a relatively small share of the total company results. Although the business segment information is a better representation of the results for copper than the total company results, the business segments contain other products in addition to copper. Therefore, conclusions must be drawn accordingly. The table shows that there is considerable variation in results, both within a company from one year to the next, as well as from one company to the next. The 5-year average ranges from a loss of 3.6 percent to a high of 13.8 percent. For ASARCO the average is 11.0 percent.

Table 7-8 shows the control costs for the smelter and the maximum percent reduction in the profit margin. This table assumes the smelter is viewed as part of a fully integrated operation. Two profit levels are shown and two capacity utilization rates (100 percent and 80 percent). The first profit level is based on a refined copper price of 187¢/kg and a 10 percent profit margin, which yields a profit of 18.7¢/kg. The second profit level is based on an increased price of refined copper to a level of 220¢/kg. The second profit margin is based on the original 18.7¢/kg, but adds the increase in price as extra profit while process costs are held constant. The second profit margin is 51.7¢/kg. Three considerations suggest the use of the second profit margin. The first consideration is the desirability of presenting sensitivity analysis in general. The second consideration is that a profit margin of 51.7¢/kg based on a price of 220¢/kg is a margin of 23.5 percent, which though clearly high, has been achieved within recent years by a member of the industry. Finally, because the margin is high, it in effect can be viewed as an upper limit, and thus if the smelter has a substantial profit impact in spite of such a favorable profit margin, it is in a very vulnerable position at a lower, more likely, profit margin. At the first profit margin (18.7¢/kg) the results show a moderate profit reduction of 8.6 percent and 10.7 percent, for the 100 and 80 percent capacity utilization rates, respectively. Profit reductions of these amounts are not likely to call into question the continued viability of the smelter. At the second, higher profit margin (51.7¢/kg) the profit impacts are lessened substantially. The profit reductions are 3.1

Table 7-8. COST INCREASE AND MAXIMUM PERCENT PROFIT DECREASE OF ARSENIC CONTROLS FOR THE HIGH-ARSENIC PRIMARY COPPER SMELTER

| | | Cost Increase (¢/kg) | | Maximum Percent Profit Decrease | | | |
|---|---|---|---|---|---|---|---|
| | | | | 18.7 ¢/kg | | 51.7 ¢/kg | |
| Smelter | Annual Capacity (Mg) | 100% Capacity Alternative II | 80% Capacity Alternative II | 100% Capacity II | 80% Capacity II | 100% Capacity II | 80% Capacity II |
| ASARCO-Tacoma | 91,000 | 1.6 | 2.0 | 8.6 | 10.7 | 3.1 | 3.9 |

percent and 3.9 percent, respectively, and do not jeopardize the viability of the smelter.

7.2.5.2 Method Two. Method two presents a supplement to method one, which introduces several additional factors that may influence an assessment of the viability of the smelter.

1) The ASARCO Tacoma smelter's ability to treat high arsenic content concentrates is unique among domestic smelters.

2) The Tacoma smelter has an economic interrelationship with ASARCO's East Helena operation. ASARCO owns and operates a lead smelter at East Helena, Montana. Matte and speiss from East Helena are shipped to Tacoma for treatment and recovery of byproducts. Therefore, the presence of the Tacoma smelter provides an economic benefit for the East Helena operation. This topic, and others, are discussed in detail in a comprehensive report on the Tacoma smelter.[36]

3) If the Tacoma smelter were to close, the remaining capacity at the Hayden and El Paso smelters would be insufficient to offset the loss and allow any significant increase in blister production. For example, disregarding any technical problems, if the Tacoma smelter were to close and the concentrates that otherwise would have gone to Tacoma were diverted to Hayden and El Paso, in 1981 those two smelters would have had to operate at a 96 percent capacity utilization rate to maintain Asarco's total blister production. A 96 percent capacity utilization rate would essentially leave no room for growth.[37]. Also, the Tacoma smelter provides an important percentage (about 21 percent) of the blister supplies for the Amarillo refinery.

4) The recent opening of ASARCO's Troy mine in Montana should be an additional long term source of concentrates for Tacoma. The Troy mine and concentrator began commercial production in late 1981 and reached full design capacity in February 1982. The Troy mine is expected to produce about 18,000 Mg of copper per year, plus a significant amount of silver.[38] A production rate of 18,000 Mg of copper per year represents a considerable amount of Tacoma's capacity, about 20 percent. With production at full capacity the ore reserves would indicate a mine life of approximately 20 years.

### 7.2.6 Capital Availability

The principal determinant of the financial viability of a smelter is profitability. However, the amount of capital needed to purchase control equipment is one of the components that enters into an evaluation of profitability. Most firms prefer to finance pollution control equipment with debt, both because debt is less expensive than equity in general, and additionally because debt incurred to purchase pollution control equipment is often tax exempt. Assuming control equipment is financed with debt, as the capital cost of the control equipment increases, the level of debt increases. An increased debt level means the fixed costs required to service the debt increase and therefore the level of risk increases. As a result, a discussion of capital availability will serve to supplement an assessment of profitability.

Table 7-9 shows the capital expenditures that will be necessary. The capital expenditures were explained in detail in an earlier chapter. The baseline capital expenditures are presented, as well as the incremental capital expenditures for Alternatives II, III, and IV. Alternatives II, III, and IV refer to the alternatives for low-arsenic smelters. Asarco owns three smelters and therefore the total capital costs are shown, although the firm can make capital budgeting decisions on an individual smelter basis. The El Paso and Hayden smelters are in the low-arsenic category, but are included here to identify total capital costs to ASARCO. The capital costs for the smelters are not trivial sums. However, the company is a major corporation. Five of the seven companies in the industry are owned wholly, or to a substantial degree, by significantly larger parent corporations. ASARCO is one of the two companies not owned by another corporation.

Table 7-9 shows the percent increase in long-term debt if controls are added. The pre-control debt level is based on a 3-year average (1981 to 1979) debt level for the company. Controls are assumed to be financed totally with debt. The baseline percentage increase in debt is 24 percent. An increase of 24 percent is substantial. For Alternatives II, III, and IV the incremental increases do not present a major obstacle. The increases are 1, 2, and 1 percent, respectively.

An additional indicator of capital availability is provided by the debt rating assigned to a company by one of the major national rating services. In 1982, as well as 1981 and 1980, ASARCO's debt was rated as A3 by

## Table 7-9. CAPITAL COSTS OF ARSENIC CONTROLS FOR ASARCO PRIMARY COPPER SMELTERS

($\$10^3$)

| Smelter | Baseline | Alternative II | Alternative III | Alternative IV |
|---|---|---|---|---|
| El Paso | 46 | 0 | 1,375 | 370 |
| Hayden | 75,606 | 0 | 1,702 | 0 |
| Tacoma | | 3,469 | 3,469 | 3,469 |
| | 75,652 | 3,469 | 6,546 | 3,839 |
| Debt Increase[a] | 24% | 1% | 2% | 1% |

[a] Percent increase in average long-term debt level for past 3-years (1981 to 1979) if controls are added as debt.

Moody's.[39] This is an investment grade rating, but it is the lowest A rating. Although an A3 rating is a relatively strong rating, it does not preclude the possibility that a substantial increase in the amount of debt by the company may present some difficulties.

## 7.3 SOCIO-ECONOMIC IMPACT ASSESSMENT

### 7.3.1 Executive Order 12291

The purpose of Section 7.3.1 is to address those tests of macroeconomic impact as presented in Executive Order 12291, and, more generally, to assess any other significant macroeconomic impacts that may result from the NESHAP. Executive Order 12291 stipulates as "major rules" those that are projected to have any of the following impacts:

- An annual effect on the economy of $100 million or more.
- A major increase in costs or prices for consumers; individual industries; Federal, State, or local government agencies; or geographic regions.
- Significant adverse effects on competition, employment, investment, productivity, innovation, or on the ability of U.S.-based enterprises to compete with foreign-based enterprises in domestic or export markets.

7.3.1.1 Annualized Control Costs. The EPA criterion for a major rule based on costs is annualized control costs of $100 million or more in any one of the first 5 years after promulgation of the standard. The annualized control cost for the ASARCO-Tacoma smelter is $1.1 million, well below $100 million.

7.3.1.2 Industry Production. The ASARCO-Tacoma smelter accounts for about 5 percent of the total U.S. primary copper smelting capacity.

7.3.1.3 Employment and Local Effects. A study performed for the City of Tacoma shows that, as of December 1977, the ASARCO Tacoma smelter and refinery employed 798 people.[40] The total employment as of December, 1976 was 866 people. The refinery has since closed, in 1980. More recently, employment at the smelter is reported at about 500 employees in 1982.[41] Related to these employment levels are indirect jobs created by the economic activity at the smelter. Those jobs numbered 478 in December 1977, and 519 in December 1976, which amounts to a total employment multiplier of 1.67. The total direct and indirect employment of the plant's operation comprises approximately 1 percent of the civilian employment in Pierce County, Washington.

Further, according to the report, ASARCO-Tacoma does business with approximately 100 local firms. The value of purchased goods and services from these local firms amounts to a little over $20 million per year. ASARCO-Tacoma paid approximately $2,370,000 in local and State taxes in 1977, and $2,245,000 in 1976. The smelter also paid $73,000 in taxes to the City of Tacoma in 1977, and $85,000 in 1976. The town of Rustin received approximately 65 percent of its general fund budget in 1977 from ASARCO.

### 7.3.2 Regulatory Flexibility

The Regulatory Flexibility Act of 1980 (RFA) requires that differential impacts of Federal regulations upon small business be identified and analyzed. The RFA stipulates that an analysis is required if a substantial number of small businesses will experience significant impacts. Both measures must be met, substantial numbers of small businesses and significant impacts, to require an analysis. If either measure is not met then no analysis is required. The EPA definition of a substantial number of small businesses in an industry is 20 percent. The EPA definition of significant impact involves three tests, as follows: one, prices for small entities rise 5 percent or more, assuming costs are not passed onto consumers; or two, annualized investment costs for pollution control are greater than 20 percent of total capital spending; or three, costs as a percent of sales for small entities are 10 percent greater than costs as a percent of sales for large entities.

The Small Business Administration (SBA) definition of a small business for Standard Industrial Classification (SIC) Code 3331, Primary Smelting and Refining of Copper is, 1,000 employees. Total employment for ASARCO is reported as 12,500 in 1981.[42] Therefore, ASARCO exceeds the SBA definition of a small business and thus no regulatory flexibility analysis is required.

## 7.4 References

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# APPENDIX A

# EVOLUTION OF THE BACKGROUND INFORMATION DOCUMENT

## EVOLUTION OF THE BACKGROUND INFORMATION DOCUMENT

| Date | Activity |
|---|---|
| July 13-14, 1976 | Emission source testing at Phelps Dodge Copper Smelter, Ajo, Arizona. |
| December 10-16, 1976 | Emission source testing at Kennecott Copper Smelter, Hayden, Arizona. |
| April 18-26, 1977 | Emission source testing at Anaconda Copper Smelter, Anaconda, Montana. |
| June 20-30, 1977 | Emission source testing at ASARCO Copper Smelter, El Paso, Texas. |
| January 17-27, 1978 | Emission source testing at ASARCO Copper Smelter, El Paso, Texas. |
| May 1-5, 1978 | Emission source testing at Phelps Dodge Copper Smelter, Douglas, Arizona. |
| May 10-12, 1978 | Emission source testing at Phelps Dodge Copper Smelter, Ajo, Arizona. |
| June 12-16, 1978 | Emission source testing at Phelps Dodge Copper Smelter, Ajo, Arizona. |
| July 11-12, 1978 | NAPCTAC Meeting in Raleigh, North Carolina to discuss issues related to development of arsenic standards for primary copper smelters. |
| July 24-27, 1978 | Emission source testing at Phelps Dodge Copper Smelter, Hidalgo, Arizona. |
| September 12-25, 1978 | Emission source testing at ASARCO Copper Smelter, Tacoma, Washington. |
| October 30 - November 15, 1978 | Emission source testing at Kennecott Copper Smelter, Garfield, Utah. |
| May 8-15, 1979 | Emission source testing at ASARCO Copper Smelter, Tacoma, Washington. |
| July 23-24, 1979 | Emission source testing at Phelps Dodge Copper Smelter, Ajo, Arizona. |

| Date | Activity |
|---|---|
| September 10-16, 1979 | Emission source testing at Phelps Dodge Copper Smelter, Morenci, Arizona. |
| September 18-22, 1979 | Emission source testing at Phelps Dodge Copper Smelter, Douglas, Arizona. |
| December 7-13, 1979 | Emission source testing at Kennecott Copper Smelter, McGill, Nevada. |
| March 11-13, 1980 | Plant visit to Hibi Kyodo Copper Smelter, Tamano, Japan. |
| April 14-23, 1980 | Emission source testing at Magma Copper Smelter, San Manuel, Arizona. |
| June 5, 1980 | EPA listing of inorganic arsenic as a hazardous pollutant under Section 112 of the Clean Air Act. |
| June 24-26, 1980 | Plant visit and emission source testing at ASARCO Copper Smelter, Tacoma, Washington. |
| March 17, 1981 | NAPCTAC meeting in Raleigh, North Carolina to discuss regulatory alternatives for limiting arsenic emissions from high arsenic throughput primary copper smelters. |
| January 12, 1983 | Judicial Opinion and Order, filed with United States District Court, Southern District of New York, pertinent to action brought by State of New York against EPA Administrator (New York v. Gorsuch, _______ F. ______ (S.D.N.Y. 1983)). Order for EPA to propose emission standards for inorganic arsenic within 180 days of Order. |
| January 14-22, 1983 | Emission source testing at ASARCO Copper Smelter, Tacoma, Washington. |
| April 27, 1983 | NAPCTAC meeting in Raleigh, North Carolina to discuss general approach and EPA staff recommendations for setting standards for inorganic arsenic emissions from primary copper smelters. |

# APPENDIX B

# INDEX TO ENVIRONMENTAL IMPACT CONSIDERATIONS

# INDEX TO ENVIRONMENTAL IMPACT CONSIDERATIONS

This appendix consists of a reference system which is cross indexed with the October 21, 1974, Federal Register (39 FR 37419) containing the Agency guidelines for the preparation of Environmental Impact Statements. This index can be used to identify sections of the document which contain data and information germane to any portion of the Federal Register guidelines.

## INDEX TO ENVIRONMENTAL IMPACT CONSIDERATIONS

| Agency Guidelines for Preparing Regulatory Action Environmental Impact Statements (39 FR 37419) | Location Within the Background Information Document (BID) |
|---|---|
| 1. Background and Description | |
| Summary of the Regulatory Alternatives | The regulatory alternatives are summarized in Section 1.2. |
| Statutory Authority | Statutory authority is cited in Section 1.1. |
| Industry Affected | A description of the industry to be affected is given in Section 7.1. |
| Sources Affected | Descriptions of the various sources to be affected are given in Section 2.2. |
| Availability of Control Technology | Information on the availability of control technology is given in Section 3.0. |
| 2. Regulatory Alternatives | |
| Regulatory Alternative I -- No Action (Baseline) | |
| Environmental Impacts | Environmental effects of Regulatory Alternative I are considered in Section 5.0. |
| Costs | Costs associated with Regulatory Alternative I are considered in Section 6.0. |

(Continued)

INDEX TO ENVIRONMENTAL IMPACT CONSIDERATIONS (Concluded)

| Agency Guidelines for Preparing Regulatory Action Environmental Impact Statements (39 FR 37419) | Location Within the Background Information Document (BID) |
|---|---|
| Regulatory Alternative II -- | |
| Environmental Impacts | Environmental effects associated with Regulatory Alternative II emission control systems are considered in Section 5.0. |
| Costs | The cost impact of Regulatory Alternative II emission control systems is considered in Section 6.0. |
| Regulatory Alternative III -- | |
| Environmental Impacts | The implementation of Regulatory Alternative III would require elimination of all arsenic emissions from the ASARCO-Tacoma smelter. This could not be accomplished without closure of the smelter. The economic impact of closure of the smelter is considered in Section 7.0. |

# APPENDIX C

# SUMMARY OF TEST DATA

# APPENDIX C
# SUMMARY OF TEST DATA

An emission test program was undertaken by EPA to evaluate the performance of alternative control techniques available for the control of process and fugitive arsenic emissions from process facilities including roasters, smelting furnaces, and converters at primary copper smelters. This appendix presents a brief description of the process facilities and control equipment tested, and a summary of the results obtained.

Arsenic emission measurements were conducted at eight domestic smelters. Particulate emission measurements were also conducted at some of these smelters. A listing of the process facilities and air pollution control equipment tested and the emission measurements conducted is presented in Table C-1. All arsenic measurements were performed using EPA Method 108, the recommended EPA method for the determination of arsenic from stationary sources. Measurements of total particulate were performed in accordance with EPA Method 5.

In addition to the arsenic and particulate emission measurements, visible emission observations were recorded at one of the above eight domestic smelters, ASARCO-Tacoma, and one other smelter located in Japan, the Tamano smelter.

A brief description of each smelter, as well as the process and control device tested, is presented in Sections C.1 through C.9. Section C.10 contains the data tables from the emissions measurement and opacity observation tests.

## C.1 ASARCO-TACOMA

The ASARCO smelter at Tacoma, Washington, is a custom smelter which processes copper ore concentrates, precipitates, and smelter by-products from numerous domestic and foreign sources. The smelter produces about 320 Mg (352 tons) of anode copper per day at full smelt and houses the only arsenic production facility in the United States. Copper smelter facilities include 10 roasters (6 Herreschoff multi-hearth roasters and 4 C&W Roasters), 2 reverberatory smelting furnaces, 4 Pierce-Smith converters, 3 anode furnaces (1 hearth type furnace and 3 tilting type furnaces),* and an electrolytic refinery.** Arsenic production facilities consist of three Godfrey roasters, arsenic trioxide settling chambers or kitchens, storage facilities, and a metallic arsenic plant.

The roaster charge which consists of a blend of concentrates, precipitates, lead speiss, flue dust, and fluxing materials typically contains 3 to 4 percent arsenic and 7 to 10 percent moisture. At full smelt, four to five roasters are used. Charging is continuous. The calcine produced, about 45.4 Mg (50 tons) per hour, is intermittently discharged from hoppers located below the roasters into larry cars for transport to one of two reverberatory furnaces. Typically, two 5.9 Mg (6.5 ton) cars are charged every 15 minutes. Fugitive emissions which could escape are confined and captured by close-fitting exhaust hoods located at the discharge point and are vented into the main roaster flue by two 73.6 $Nm^3$/min (2,600 scfm) fans. In addition, general ventilation is applied at the south end of the larry car tunnel to control fugitive emissions resulting from the re-entrainment of settled dust within the tunnel.

Offgases from the roasters, which average about 3,570 $m^3$/min (126,000 acfm) at 260°C (500°F), are combined with the exhaust gases from the ancillary fugitive emission control systems [850 $Nm^3$/min (30,000 scfm)] and reduced in temperature to less than 120°C (250°F). The total gases are then treated in a baghouse for particulate removal. The baghouse consists of 17 compartments containing 120 bags each. The bags are made of acrylic and measure 20.3 cm (8 inches) in diameter and 7.6 meters (25 feet) in length. The total baghouse filtering area is 9,950 $m^2$ (107,100 $ft^2$). The baghouse is designed to effectively treat 5,664 $m^3$/min (200,000 acfm) of gas at an air-to-cloth ratio of about 1.9 to 1.0. Bag cleaning is performed by mechanical shakers. The clean baghouse exhaust is vented through a flue to the smelter main stack.

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* Starting September 1978 only the two tilting furnaces are being used; the hearth type furnace is no longer used.

**Operation of electrolytic refinery was discontinued in January 1979.

Although the smelter has two reverberatory smelting furnaces, each with an approximate smelting capacity of 1,090 Mg (1,200 tons) per day, the designated Number 2 furnace is used almost exclusively. The furnace measures 33.5 m (110 feet) in length and 9.8 m (32 feet) in width and is fired by either oil or natural gas. Furnace charging is accomplished by discharging the larry cars through one of four Wagstaff guns located along the furnace sidewalls. Typically, it takes less than 1 minute to discharge each car. At full smelt, two cars are discharged about every 15 minutes. To minimize potential fugitive emissions during charging, a manual control override is used which simultaneously opens the furnace flue control damper and reduces the fuel supply to the furnace fire prior to each charge to prevent pressure surges in the furnace.

Matte is tapped from the furnace as required by converter operations through one of four tapping ports. Only one tapping port is used at a time. The matte flows through a cast copper launder to a 4.25 $m^3$ (150 $ft^3$) cast steel ladle for transfer to the converters. At full smelt, about 45 ladles are transferred per day. It takes about 7 minutes to fill one ladle. Similarly, slag is skimmed through one of two tapping ports as required to maintain the proper slag level in the furnace and transferred to a 5-pot slag train for transit to the slag dump. Once tapped, the slag flows through a cast steel launder and into a 2.8 $m^3$ (100 $ft^3$) cast steel slag pot. At full smelt, about 20 five-pot slag trains are dumped per day. Each train takes about 15 minutes to fill. Fugitive emissions generated during both matte and slag tapping operations are controlled by local ventilation techniques. Tapping ports, launders, and launder-to-ladle/pot transfer points are hooded and ventilated. Ventilation requirements for the matte tapping system total about 700 $m^3$/min (25,000 acfm), while the ventilation requirements for the slag skimming system total about 600 $m^3$/min

(21,000 acfm). Captured emissions from both systems are currently controlled by an electrostatic precipitator prior to venting from a stack to the atmosphere.

Process gases from the reverberatory furnace, which average about 1,415 $Nm^3$/min (50,000 scfm), pass through a pair of waste heat boilers where the gases are cooled to about 400°C (750°F). The exiting gases then pass through a large rectangular brick flue where additional cooling and gas stream conditioning is provided by air infiltration and water and sulfuric acid sprays located in the flue. The resultant gas stream, about 6,100 actual $m^3$/min (215,000 acfm) at 132°C (270°F), then enters the first of two electrostatic precipitators in series for particulate removal. The first precipitator is a tube or pipe design consisting of 18 sections with a total collection area of about 6,619 $m^2$ (71,250 $ft^2$). Each section contains 84 pipes measuring 30.5 cm (12 inches) in diameter and 4.6 m (15 feet) in length. The second unit is a plate type design. It consists of seven parallel chambers each with four fields in series and has a total collection area of 7,710 $m^2$ (82,992 $ft^2$). The exiting gases, about 7,740 actual $m^3$/min (270,000 acfm) at 110°C (230°F), are discharged through a large flue to the smelter main stack.

Matte from the reverberatory furnace is transferred to one of four Pierce-Smith converters. Three of the converters measure 4.0 m (13 feet) in diameter by 9.1 m (30 feet) in length, while the fourth converter is 3.4 m (11 feet) in diameter and 7.9 m (26 feet) long. In addition to copper matte, smelter reverts and cold dope materials are also processed. Typically, only two converters are on blow at any one time. A converter cycle normally takes from 10 to 12 hours. With dilution air, the offgas flow per blowing converter is about 1,130 $Nm^3$/min (40,000 scfm) and contains from 3 to 4 percent $SO_2$. Blister copper produced is transferred to one of three anode furnaces for refining and casting. The slag skimmed from the converters is recycled to the reverberatory furnace.

Offgases from converter blowing operations are captured by water-cooled hoods and pass through a series of multiclones and a settling flue for coarse particulate removal prior to entering the gas cleaning circuits of either a liquid $SO_2$ plant or single-contact sulfuric acid plant. The gas cleaning circuits for both plants are similar and consist of a water spray chamber, electrostatic precipitator, scrubbers, and mist precipitators in series. The single-contact acid plant has a 182 Mg/day (200 TPD) capacity at 5 percent $SO_2$ and is capable of processing 652 $Nm^3$/min (23,000 scfm) of converter gas. The liquid $SO_2$ plant processes up to 12,748 $Nm^3$/min (45,000 scfm) of the converter gases. The plant uses dimethylaniline (DMA) to absorb the $SO_2$ in the gas stream and uses steam stripping from regeneration. The 100 percent concentrated $SO_2$ gas stream produced is then liquified by compression and the liquid $SO_2$ stored.

Inlet and outlet emission measurements for arsenic were conducted by EPA across the roaster baghouse and the arsenic baghouse (metallic and kitchen) on September 12 through September 25, 1978. Arsenic test results are summarized in Tables C-3 through C-7.

Testing for arsenic was also conducted on September 15, 16, and 18, 1978, at the outlet of the reverberatory furnace ESP. These results are shown in Table C-8.

Tests were also conducted for roaster calcine discharge, matte tapping, slag tapping, and converter slag return. Data obtained for these sources are presented in Tables C-9 through C-12.

Visible emission observations were made by using EPA Method 22 and EPA Method 9 for calcine loading of larry cars, matte tapping, slag tapping, and converter slag return on June 24-26, 1980. Tables C-65 through C-73 present the results of these visible emissions observations.

## C.2 ASARCO-EL PASO

The ASARCO smelter at El Paso is a custom smelter that processes copper ore concentrates from numerous sources. The smelter produces about 315 Mg (350 tons) of anode copper per day. Copper smelting facilities consist of ore handling and bedding facilities, four multi-hearth roasters, one reverberatory smelting furnace, and three Pierce-Smith converters. In addition, this smelter also has separate process facilities for zinc and lead production.

The smelter feed, consisting of a blend of concentrates, precipitates, lime and flue dust, and typically containing more than 0.2 percent arsenic, is charged to four Herreshoff multi-hearth roasters. Each roaster has seven hearths and is capable of producing approximately 356 Mg (392 tons) of calcine per day. The calcine is taken by larry car to the single reverberatory furnace which is 90 cm (35.7 in.) long and 20 cm (8.0 in.) wide and fired with either oil or natural gas. The furnace is charged by Wagstaff guns located along the furnace side walls. The matte is tapped from one of five tap holes located on the north and south sides of the furnace. Slag is tapped from the west side of the furnace and is disposed of in a slag dump.

Matte from the furnace is transported to one of three Pierce-Smith converters. Each of the converters is 4 m in diameter and 9 m long. Normally, two converters are in the blowing mode at one time. In addition to copper matte, flue dust and cold dope materials, converters also process lead matte from the adjacent lead smelter when available. Blowing time generally ranges from 10 to 12 hours. Blister copper produced is further refined in two anode furnaces and then cast into anodes for shipment.

Offgases from the reverberatory furnace, which average about 1,700 $Nm^3$/min (60,000 scfm), pass through a pair of waste heat boilers where the gases are cooled to about 400°C (750°F) and about 23 Mg (50,000 lb) of steam is produced. The gases exit the waste heat boilers through two parallel ducts and are then combined with the roaster offgases in a main flue. The combined gas stream, consisting of about 5,000 $Nm^3$/min (177,000 scfm) at 200°C (400°F), then passes through a spray chamber where it is cooled to about 110°C (230°F) prior to entering an electrostatic precipitator for particulate removal. The precipitator consists of seven parallel chambers. Each chamber has four fields in series and has a total field volume of 535 $m^3$ (18,900 $ft^3$). Gases exiting the precipitator are discharged to the main stack through a large balloon flue.

Offgases from the converter blowing operations average 6,000 $Nm^3$/min (56,500 scfm). They then pass through a settling chamber, two waste heat boilers, and a spray chamber where the gases are cooled from 315°C (600°F) to 110°C (230°F). This process occurs prior to treatment in an electrostatic precipitator for particulates.

The precipitator consists of four parallel chambers, each of which has four fields in series. The exiting offgases then pass through a venturi scrubber for additional particulate removal, are humidified and cooled in a pair of packed-bed scrubbers, and are treated in a series of mist precipitators where acid mist and any remaining particulates are removed prior to entering a double-contact acid plant for $SO_2$ removal. The acid plant has a normal production rate of 408 Mg (450 tons) of acid per day. Either 93 or 98 percent sulfuric acid is produced. The acid plant tail gas streams are discharged through a 30.5 m (100 ft) stack.

Emission measurements were conducted by EPA during June 26-30, 1977. Inlet and outlet arsenic and mass measurements were made across the roaster/reverberatory electrostatic precipitator. Three inlet locations and one outlet location were sampled. The inlet locations included a large downtake duct off of the multi-hearth roasters, and two parallel ducts downstream of the reverberatory waste heat boilers. The outlet location consisted of the balloon flue downstream of the precipitator. Three arsenic and two total particulate runs were conducted. The arsenic and particulate test results are summarized in Tables C-13 through C-22, and Table C-29.

Arsenic emission tests were also conducted across the double-contact sulfuric acid plant. Three inlet and outlet measurements were made. The sampling locations included a duct upstream of the spray chamber/ESP and the acid plant tail gas stack. The results of these tests are summarized in Tables C-23 and C-24. Process conditions were carefully observed, and testing was conducted only when the subject process facilities and control equipment were operating within normal operating limits.

Fugitive emissions from the reverberatory furnace matte tapping operation at ASARCO-El Paso are captured by hoods over the ladle, covered matte launders, and a hood at the matte tapping holes. Gases from these sources are combined in a common duct and directed through a fan to a baghouse. The baghouse discharges into the roaster/reverberatory spray chamber/ESP control system which discharges from the 250 m (828 ft) main stack.

Fugitive gases that escape the converters during the blow period, and roll-in/out operations and other fugitive gases in the converter building, are collected at the roof of the converter building. Collected fugitive gases are drawn through four openings at the roof into ducts that combine into a main duct leading to a baghouse, through fans on the clean side of the baghouse, and then out the 250 m (828 ft) main stack.

The fugitive gas flow through the baghouse averages 14,100 $Nm^3$/min (498,000 scfm). The converter building fugitive baghouse consists of 12 compartments. Normally all compartments are in operation except one compartment which is taken off during the cleaning cycle and another compartment which is taken off for maintenance purposes for a fraction of the total time. Each of the 12 compartments contains 384 Orlon or Dacron bags. Each compartment is 20 cm (8 in.) in diameter, 6.7 m (22 ft) long, with a cloth area of 1,644 $m^2$ (17,700 $ft^2$) per compartment. The total net cloth area of the baghouse is about 19,700 $m^2$ (212,400 $ft^2$). The baghouse was designed to effectively treat 15,282 actual $m^3$/min (540,000 acfm) at 54°C (130°F) using an air-to-cloth ratio of 3.0 to 1.0. Mechanical shakers (automatic) are used for cleaning. Dust from the baghouse is removed from the dust chambers under the baghouse by screw conveyors.

Emission measurements across the baghouse were conducted by EPA during Janaury 17-27, 1978. Three arsenic and particulate measurements were made at the inlet and outlet locations of the converter building's fugitive baghouse. The arsenic test results are summarized in Tables C-25 and C-26, and the particulate test results are summarized in Tables C-27 and C-28. During the same period, three arsenic and particulate measurements were made at the inlet to the matte tapping baghouse and the calcine discharge duct. Due to the physical configuration of the matte tapping baghouse system, outlet tests could not be conducted. Measurements were conducted after the fan; however, a side stream of about 311 actual $m^3$/min (11,000 acfm) of the matte tapping gases were split at the fan and ducted to the reverberatory furnace waste heat boilers for cooling purposes. This gas stream was measured only for volume flow. It was assumed that the pollutants in this gas stream would have the same concentration as the gas stream measured going to the baghouse. Process conditions and control device parameters (when applicable) were carefully observed during all the periods when testing was conducted. Tests were conducted only when the process facilities and control equipment were operating within normal limits. Uncontrolled arsenic test results are summarized in Tables C-30 and C-32, and particulate tests are given in Tables C-31 and C-33.

## C.3 ANACONDA

This smelter, when operating, was producing about 545 Mg (600 tons) of anode copper daily. Major process facilities consisted of a fluid-bed roaster, electric smelting furnace, six converters (three operational), and an anode furnace.

In this process, concentrates and precipitates are blended with silica flux in approximate proportions of 88, 2, and 10 percent, respectively. The blended materials (containing about 0.96 percent arsenic) are then fed to a Dorr-Oliver designed fluid-bed roaster by a screw feeder which controls the feed rate [typically about 91 Mg (100 tons) per hour] and maintains a seal on the roaster. Fluidizing air averages 1,062 $Nm^3$/min (37,500 scfm). The air is supplied through tuyeres at the bottom of the roaster to keep the bed constantly fluidized at 1.8 m (70 in.) in depth.

The fluidized air reacts with the sulfur contained in the sulfide ores to form $SO_2$ and calcine. Approximately 45 percent of the sulfur contained in the feed material is eliminated. Because the reaction is exothermic, no auxiliary fuel is needed except at cold startup. The bed temperature is generally maintained at 582°C (1,080°F). Most of the calcine which is produced (85 percent) exits the reactor as a fine dust suspended in the offgas stream. The offgases average 1,671 $Nm^3$/min (59,000 scfm) at 543°C (1,010°F). These gases are ducted through a series of primary and secondary cyclones where 90 to 95 percent of the suspended calcine is recovered. The underflow from the roaster accounts for the remaining 15 percent of the calcine produced.

An electric furnace is used for smelting. Drag conveyors continuously distribute calcine produced in the roaster in combination with some recycled flue dusts, along each side of the furnace through a series of charge pipes in the roof. The furnace working area is 18 m wide, 36 m long, and 30 m high. The hath area is 272 $m^2$. The furnace is equipped with six carbon electrodes. Each is 1.65 m in diameter. The electrodes are energized by three transformers. Each has a capacity of 12 MVA. Normal voltage applied is between 150 and 180 volts. The furnace production capacity is 998 Mg (1,100 tons) of matte per day at 52 percent copper. Matte and slag are tapped as required from four matte and two slag tap holes to maintain normal depths (matte, 71 to 96 cm; slag, 102 to 152 cm). Offgases from the furnace average approximately 425 $Nm^3$/min (15,000 scfm) at 645°C (1,200°F).

Matte from the electric furnace is transported to one of three Pierce-Smith converters. Typical converter process feed rates include 41.5 Mg (45.7 tons) of matte, 4.5 Mg (5.0 tons) of flux, and 1.4 Mg (1.5 tons) of cold dope per hour. The offgas flow is about 2,832 $Nm^3$/min (100,000 scfm) per blowing converter because of excessive air infiltration and typically contains about 2.5 percent $SO_2$. Blister copper which is produced is transported to an anode furnace for further refining and subsequently poured into anodes on a casting wheel.

Process offgases from the electric furnace and converter blowing operations are combined. Approximately 2,070 $Nm^3$/min (73,000 scfm) of the resultant gas stream is treated in a cooling chamber (humidified and cooled) and a venturi scrubber for particulate removal prior to entering a 600 Mg (design) double-contact acid plant. The remaining gases are combined with the fluid-bed roaster offgases and transported through a large balloon flue to a spray chamber/baghouse filtration plant for particulate control.

The combined gas stream, consisting of roaster, electric furnace, and converter process gases totaling about 5,664 $Nm^3$/min (200,000 scfm) and ranging from 230 to 340°C (450 to 650°F) in temperature, exits the balloon flue and enters two parallel spray chambers where the gases are cooled to less than 110°C (230°F) with watersprays. Each spray chamber is 7.3 m wide, 4.3 m high, and 30 m long and is equipped with 10 sonic spray nozzles. Water requirements range from 265 to 303 liters/min (70 to 80 gpm). The cooled gas stream then passes through two of three fans (one is standby), each with a capacity of 6,230 actual $Nm^3$/min (220,000 scfm), prior to entering the baghouse.

The baghouse consists of 18 compartments aligned in two rows of nine. Each compartment is 4.3 by 14.3 m in cross section and 11.3 m high above the thimble flow. The 18 chambers are constructed of reinforced concrete above the thimble flow and are completely insulated. Each of the 18 compartments contains 240 Orlon bags which are 3.5 m in diameter and 7.7 m in length. The cloth area per compartment is 1,656 $m^2$. The baghouse net cloth area totals about 29,802 $m^2$. The baghouse is designed to effectively treat 11,328 actual $Nm^3$/min (400,000 acfm) at an air-to-cloth ratio of 1.25 to 1.00. Mechanical shakers are used for cleaning. The clean baghouse exhaust is transported via a high velocity insulated fiberglass flue to the base of the main stack and subsequently discharged to the atmosphere.

Inlet and outlet emission measurements for both arsenic and particulates were conducted by EPA across the spray chamber/baghouse on April 20-26, 1977. Sampling was conducted upstream of the spray chamber and downstream of the baghouse. Two sampling locations were required to obtain the inlet values. During all tests, process conditions were closely monitored, and testing was conducted only when the process facilities were operating within normal operating limits.

The arsenic and particulate test results for the spray chamber/baghouse are summarized in Tables C-34 through C-41.

## C.4 PHELPS DODGE-AJO

This is a "green" feed smelter with a production capacity of about 168 Mg (185 tons) of anode copper per day. Major process facilities consist of a single reverberatory furnace, three Pierce-Smith converters, and oxidizing furnace, and an anode furnace. Emission control apparatus includes electrostatic precipitators for the control of particulate emissions from smelting and converting operations and a single-contact acid plant.

The reverberatory furnace, which is designed for wet smelting, is 9 m (30 ft) wide and 30 m (100 ft) long. It is fired with natural gas or fuel oil, depending on the availability of gas. The furnace walls and roof are constructed of silica brick, and the roof is of a sprung-arch design. The furnace charge components consist of concentrates (90 percent), precipitates, limestone, and recycled flue dusts. In addition, converter slag is returned and processed. A charge, usually 1.8 to 3.6 Mg (2 to 4 tons), enters the furnace through one of six charge ports, three on each side of the furnace. Each port is equipped with a high-speed belt slinger to charge wet concentrates at considerable velocity. Slag and matte are tapped as required to maintain a normal bath depth of approximately 120 cm (46 in.) in the furnace.

Offgases from the reverberatory furnace pass through two parallel waste heat boilers where the gases are cooled to about 315°C (600°F), and a significant quantity of steam is produced for power generation. The exiting gas streams then enter a common plenum chamber for mixing prior to treatment in a hot electrostatic precipitator which is designed to handle 4,250 actual $m^3$/min (150,000 acfm) at 315°C (600°F). The precipitator is a Joy-Western design, which was installed in 1973. It consists of two parallel units with two stages each, and it has a total collection area of 3,860 $m^2$. Gas treatment averages 6 seconds at a gas velocity of 0.9 m (3 ft)/sec. The pressure drop across the precipitator is 1.3 cm (0.5 in.) of water maximum. The unit has a design efficiency of 96.8 percent measured at its operating temperature.

Offgases from the converters pass through waste heat boilers where gases are cooled to about 315°C (600°F), and steam is generated from the removed heat. Gases enter a balloon flue and then pass through an electrostatic precipitator (ESP). The ESP has two independent horizontal parallel units with three fields each, which are designed to handle 5,940 actual $m^3$/min (210,000 acfm) at 340°C (650°F) and 95,100 pascals (13.8 psia). Total ESP collecting surface area is 2,770 $m^2$ (29,808 $ft^2$). After the converter gases leave the ESP, they pass onto the scrubbing section of the acid plant where they are treated in a humidifying tower, a cooling tower, and a mist precipitator. The cleaned offgases are then processed in a single absorption 544 Mg/day (600 ton/day) acid plant for $SO_2$ removal. Either 93 or 98 percent sulfuric acid can be produced. The acid plant tail gas is ducted to the main smelter stack.

Simultaneous inlet and outlet arsenic emission measurements were conducted by EPA on July 13-14, 1976, on the reverberatory furnace

ESP. Operating values for primary and secondary voltage and current and spark rate were monitored during all tests to ensure normal operation. In addition, reverberatory furnace operations were monitored for normal operation. The test results are summarized in Tables C-42 and C-43.

Inlet Arsenic emission measurements at the converter ESP inlet were conducted by EPA on June 13-15, 1978. Outlet arsenic emission measurements at the inlet of the acid plant, and acid plant outlet tests, were conducted at the same time. Test results are given in Tables C-44, through C47.

Fugitive emissions at Phelps Dodge-Ajo escape the primary hooding on the converters during the blowing cycle. These emissions are captured by fixed, semicircular-shaped secondary hoods attached to each converter. The hoods are approximately 1 m (3 ft) high at the tallest point. Although the secondary hoods are in operation during the blowing cycle, they are dampered when the converters are in the roll-out mode. The secondary system is designed to handle 1,980 $Nm^3$/min (70,000 scfm) of gases. The gases are ducted uncontrolled to the main stack.

Fugitive emissions from the two matte tapping locations are controlled to a high degree by a rectangular duct about 60 to 90 cm (2 to 3 ft) above each tap hole. The rectangular vent opening is about 60 cm (24 in.) wide by 30 cm (12 in.) high and controls the fumes from the matte tapping hole and about one- to two-thirds down the matte launder which is not covered. The gas volume for this system is about 850 $Nm^3$/min (30,000 scfm) and provides considerable draft to the vent for several feet. The matte runs into a 2.4 to 3 m (8 to 10 ft) launder and drops into a ladle on the floor below. Fumes from the matte launder, are drawn into the system handling the ladle emissions. Most of the time, few emissions escape from the matte launder.

Fugitive emissions from the matte pouring into and from the ladle are captured in a cubicle where the ladle is located. The ladle is placed on a specially designed cradle on rail tracks. An electric motor and pulley arrangement moves the ladle car in and out of the cubicle. During the matte tap, the ladle is placed in the cubicle, and the doors are closed. The cubicle has ventilation ducts leading from it through a fan, and the gases are discharged through the main stack. The ventilation rate from the two cubicles is 1,700 $Nm^3$/min (60,000 scfm).

Fugitive emissions from the slag tapping location are captured by using a hood (similar to matte tapping) at the slag tapping hole. The slag pours into an open launder and drops through a hole in the floor into a slag ladle positioned below. A hood near the ladle captures fume from the ladle. Normally, few emissions emanate from the slag tapping operation. The ventilation rate for the slag tapping system is about 850 $Nm^3$/min (30,000 scfm). The duct from the slag tapping system is combined with the matte tapping duct system, and the gases are discharged out of the main stack.

Uncontrolled emission measurements for arsenic, particulate matter, and $SO_2$ were conducted by EPA for the secondary converter hood and matte tapping systems on May 10-12, 1978. During all tests, process conditions were closely monitored, and testing was conducted only when the process facilities were operating within normal limits. The arsenic test results are summarized for both systems in Tables C-48, and C-50. The particulate/ $SO_2$ test results are summarized for both systems in Tables C-49 and C-51.

## C.5 PHELPS DODGE-HIDALGO

This smelter produces about 398 Mg (438 tons) of anode copper daily. Major process facilities consist of a rotary dryer, flash furnace, electric slag furnace, three converters, and two anode furnaces.

Concentrates, fluxes, and dusts are blended in approximate proportions of 88, 10, and 2.0 percent, respectively. Blended materials (containing 0.005 to 0.06 percent arsenic) are then fed to a rotary dryer at about 88 Mg (97 tons) per hour. These materials are heated by gases passing through the steam superheater and process air preheaters. The volume of combined gas streams is about 708 $Nm^3$/min (25,000 scfm). They enter the dryer at about 315°C (600°F) and leave the dryer at about 80 to 100°C (180 to 220°F). The dryer discharges into a lift tank containing blended material and the fine dust portion of the ESP cleaning the dryer offgases. The lift tank material is charged to the dry charge bins feeding the flash furnace.

A flash furnace is used for smelting. The feed material is charged through holes in the roof on the reaction shaft side of the furnace. The furnace reaction shaft is about 8 m (27 ft) inside diameter by about 36 ft high. The settling chamber is about 23 m (76.5 ft) long, 10 m (33 ft) wide, and 6 m (20.5 ft) high. The uptake shaft is about 9 m (30 ft) long, 6.6 m (21.5 ft) wide, and 16.6 m (54.5 ft) high. These are all inside dimensions. Ambient air, preheated to 370 to 450°C (700 to 850°F), is fed to the furnace; the process offgas is about 2,430 $Nm^3$/min (86,000 scfm) at 1,200°C (2,200°F) and 10 percent $SO_2$. The furnace operating temperature is normally 1,350°C (2,460°F). Furnace production is about 715 Mg/day (650 tons/day) of matte and 1,650 Mg/day (1,500 tons/day) of slag. The slag from the flash furnace is tapped directly to the electric furnace for further copper recovery.

Matte from the flash furnace is transported to one of three Pierce-Smith converters. One converter usually is on a slag blow, one on a copper blow, and one is prepared for the next matte charges or on standby. The converter feed rates are 24 Mg (27 tons) of matte from the flash furnace, 8 Mg (9 tons) of matte from the electric furnace, and 1.7 Mg (1.9 tons) of flux per hour. The converters produce nearly 18 to 19 Mg/hr (20 to 21 tons/hr) of blister copper and 4.5 to 5.4 Mg/hr (5.0 to 6.0 tons/hr) of converter slag which is sent to the electric furnace.

An electric furnace is used for further processing of the flash furnace slag and converter slag. The furnace uses about 94 kWh per ton of charged material. The feed rates are about 66 Mg (60 tons) of flash furnace slag, 4.5 to 5.4 Mg (5.0 to 6.0 tons) of converter slag, and can handle about 11 Mg (12 tons) of reverberatories per hour. The furnace produces about 7.2 to 8.1 Mg/hr (8.0 to 9.0 tons/hr) of matte and 50 Mg/hr (tons/hr) of slag. The matte is transferred to the converters by ladle, and the slag is transported to a slag dump.

Blister copper produced in the converter is transported by ladle to one of two anode furnaces where it is further reduced to anode copper. The copper is then poured into anode molds on a casting wheel at 16 to 17 Mg/hr (18 to 19 tons/hr). The anode copper is loaded on railcars and sent to a copper refinery for further processing.

Fugitive emissions escaping the primary hooding on the converters are captured by secondary hoods designed by the company. The secondary hoods have sliding doors 8.8 m (29 ft) long, 5.8 m (19 ft) wide, and 5.5 m (18 ft) high and cover the converter to the operator floor level. The design gas volume handled by the converter fugitive system is about 2,830 $Nm^3$/min (100,000 scfm), and the temperature of the gases varies from 150 to 260°C (300 to 500°F).

Emission measurements were conducted by EPA during July 25-26, 1978, in the fugitive converter duct. The location was downstream of all the converters. Three emission measurements were made for uncontrolled arsenic. During the tests, process conditions were closely monitored, and testing was conducted only when the process facilities were operating within normal limits. The test results are summarized in Table C-52.

### C.6 PHELPS DODGE-DOUGLAS

The Douglas Reduction Works is a calcine fed smelter producing about 322 Mg (365 tons) per day of 99.6 percent copper anodes. Copper anodes are sent to a copper refinery for further processing. The major units at the smelter include 24 roasters, 3 reverberatory furnaces, 5 converters, and 2 anode furnaces.

The roaster process consists of 24, 7-hearth Herreshoff roasters arranged in two parallel rows of 12. Only 18 are normally in operation at a time. The roasters are standard Herreshoff with a shell diameter of about 6.7 m (22 ft). Natural gas is introduced near the bottom of the roasters. As the hot gases rise, feed material introduced at the top is forced down through the roasters by the use of rabble arms which spread the feed around each hearth level and through openings at each level. About 154 to 163 Mg (170 to 180 tons) of calcine per roaster is produced each day.

Calcine from the roasters is discharged into holding hoppers where it is transferred by gravity to larry cars and delivered by rail to the reverberatory furnace.

The roaster calcine discharge emission control system consists of hoods covering the larry car with flexible screening or flaps hanging from the edges of the hoods to the top of the larry car. The hoods are split open 20 cm (8 in.) in the center to allow the power pole for the electric motor driving the larry cars to make contact with the electrical source. Unfortunately, when the hoppers are discharged into the larry car, the split allows calcine dust to escape from the hood into the building. The discharge ducts from the hoods are combined into a single duct leading to cyclones, a baghouse, a fan, and then discharged through a long duct leading to the roaster/reverberatory stack discharging to the atmosphere. Each hood can be dampered so that all the gas volume (draft) is available to the hoods used for calcine discharge.

The baghouse consists of eight compartments with 180 polyester bags per compartment. The bags are 328 cm (129 in.) long and 13 cm (5.0 in.) wide. The baghouse was designed to treat 1,130 actual $m^3$/min (40,000 acfm) at ambient temperatures. Mechanical shakers are used for cleaning, and the baghouse and cyclone dusts are returned to the calcine hoppers by screw conveyors.

Inlet and outlet emission measurements for arsenic and particulates were conducted by EPA across the baghouse on May 3-5, 1978. Sampling was conducted upstream of the cyclones and downstream of the fan. During all tests, process conditions were closely monitored, and testing was conducted only when the process facilities were operating within normal operating limits.

Arsenic test results are summarized in Tables C-53 and C-54. Particulate test results are summarized in Tables C-55 and C-56.

## C.7 KENNECOTT-MAGNA, UTAH

This smelter was designed to produce nearly 680 Mg (750 tons) of anode copper daily. Major process facilities consist of two rotary concentrate dryers, three continuous Noranda smelting vessels, four converters, and four anode furnaces.

Concentrates and precipitates, dusts, and slag concentrates are blended with silica flux in approximate proportions of 78, 4, 12, and 6 percent, respectively. Feed material from storage bins are fed to conveyor belts that transport the material to a slinger (high-speed conveyor system) which charges the material into the smelting vessels. Matte and slag are normally tapped about every 30 minutes. The slag is taken to a processing station where it is cooled, crushed, and reprocessed to recover additional copper values (slag concentrate) which are returned to the smelting cycle. Matte is transported by ladle to the converters. Blister copper from the converters is transported by ladle to the anode furnaces. The strong $SO_2$ offgas streams from the smelting vessels and converters are cleaned and delivered to the acid plants.

Emissions from the matte tapping operation (hole) are captured in an enclosed cubicle. The ladle on a cradle car on rails (designed for this purpose), is in another cubicle beneath the matte tap floor level under the smelting vessel. Ducts direct the fumes from these operations into one duct which leads to the main fugitive duct system at the roof of the smelting and converter building. The main fugitive duct handles all fugitive emissions for the smelter area and discharges these fugitive emissions through a spray chamber and out the main stack (1,200 ft). The gas flow for the matte tapping hole system is 708 $Nm^3$/min (25,000 scfm) and for the ladle system is 2,120 $Nm^3$/min (75,000 scfm).

Emissions from the slag tapping operation are captured by a rectangular duct opening beside the tapping hole with a flow of 340 $Nm^3$/min (12,000 scfm). A similar rectangular duct opening captures the emissions from the slag ladle in the area beneath the end of the smelting vessel. The flow rate for the collection of these emissions is 990 $Nm^3$/min (35,000 scfm).

Emissions escaping the primary converter hoods during the blow, and some emissions escaping the converters during the roll-out mode, are captured by secondary hoods and ducted to the main fugitive duct, which was explained previously. The secondary hooding system has a steel shell constructed over the primary hoods. The primary hoods are permanent, nonmovable hood arrangements above the converters. This allows a transition to the fugitive duct system. The lower part of the secondary hood system consists of four doors that close over the converter and ladle area beneath the converter. One door turns down from the top, two doors close in the center, and the last door moves across the bottom area, which completely covers the converter. Unlike other smelters with secondary hoods, the fugitive system at this smelter operates at all times, whether the converter is in the blowing cycle or roll-out mode. The secondary converter doors were not operable during the tests due to mechanical problems with the doors. The design gas volume per converter is 2,745 actual $m^3$/min (97,000 acfm) at 82°C (180°F).

The converter process gas stream was also source tested for uncontrolled arsenic. The tests were conducted before one of three fans and the control device to the acid plants. There is a settling chamber at the converters, and also a plenum chamber with dampers where the gases from each of the converters can be directed to one, two, or all three fans (as needed) to the acid plants. The gas volume for this process stream per converter is about 425 $Nm^3$/min (15,000 scfm).

Two rotary dryers are used at this smelter to dry concentrate feed for the smelting vessels. Emissions from the dryers are controlled by a cyclone followed by a spray chamber scrubber.

Uncontrolled arsenic emission measurements were conducted by EPA on November 1-14, 1978, on the slag and matte tapping system, converter secondary hood system, and the converter process gas stream before the control system. The rotary dryer scrubber outlet was also measured for arsenic during this period. During all tests, process conditions were closely monitored, and testing was conducted only when the process facilities were operating within normal limits.

The arsenic test results for these processes are summarized in Tables C-57 through C-62.

## C.8 KENNECOTT-HAYDEN

The Kennecott Copper Corporation, Ray Mines Division, smelter at Hayden, Arizona, was originally put on stream in mid-1958 and extensively modernized in 1969 and 1973. The smelter produces nearly 227 Mg (250 tons) of anode copper daily and consists of a concentrator plant, fluid-bed roaster, reverberatory smelting furnace, three converters, two anode furnaces, an anode casting wheel, and a double-contact acid plant.

Concentrate and precipitates from the concentrator plant are blended with silica flux in proportions of 86.3, 2.2, and 11.5 percent, respectively. The blended materials (containing 6 to 12 percent moisture and less than 0.015 percent arsenic) are then fed to a Dorr-Oliver designed fluid-bed roaster by a screw feeder. The feeder controls the feed rate and maintains a seal on the roaster. The roaster feed rate typically ranges from 45.4 to 63.5 Mg/hr (50 to 70 tons/hr). Fluidizing air averages 425 $Nm^3$/min (15,000 scfm). The air is supplied through 373 tuyeres at the bottom of the reactor vessel to keep the bed (approximately 1.8 m in depth) constantly fluidized.

The fluidizing air reacts with the sulfur contained in the sulfide ores to form $SO_2$ and calcine. Approximately 50 percent of the sulfur contained in the feed material is oxidized to $SO_2$. Because the reaction is exothermic, no auxiliary fuel is needed except for cold startup. The bed temperature is generally maintained between 565 and 620°C (1,050 to 1,150°F). Most of the calcine produced (85 percent) exits the reactor as a fine dust suspended in the offgas stream. The offgases average 623 $Nm^3$/min (22,000 scfm). They are then ducted through a series of four primary and four secondary cyclones. In the cyclones, about 95 percent of the suspended calcine is recovered and subsequently conveyed by screw conveyor to the calcine storage bin. The underflow from the reactor, which accounts for about 15 percent of the calcine produced, is reclaimed through an underflow valve and transported to the calcine storage bin by a drag chain conveyor.

Calcine, precipitator dust, and flux are then fed to a single reverberatory furnace for smelting. The reverberatory furnace (a suspended arch design) is 9.1 m (30 ft) wide by 30.5 m (100 ft) long. The furnace is charged through two openings at the top by a pair of Wagstaff feeders. The furnace is charged every 15 minutes for a duration of 2 to 3 minutes. Unless actually being charged, a slight negative draft is maintained across the furnace (about -1.5 mm of $H_2O$). The furnace is fired with natural gas but is equipped with oil burners in the event gas service is interrupted. Slag from the furnace is periodically tapped into slag pots which are subsequently hauled by rail to the slag dump. About 650 tons are removed daily. Matte is tapped into ladles which are moved into the converter aisle when full.

The matte ladles are then picked up by an overhead crane and charged to one of three Pierce-Smith converters. Each is 4.0 m (13 ft) in diameter by 9.1 m (30 ft) long and equipped with 42 tuyeres. Present converter operation consists of keeping two converters on charge concurrently. Each is in the blowing cycle for 50 percent of the time for 24 hours. Air flowing at 595 $Nm^3$/min (21,000 scfm) blows through the tuyeres in the matte charge, flux added, and iron oxide slag produced. The slag is then skimmed and poured into ladles. Unlike most domestic smelters, the converter slag is not charged to the reverberatory furnace, but is carried to a special slag pit where it is cooled and subsequently returned to the concentrator and blended with raw ore.

Additional matte and dope materials (reverts or copper scrap) are added to an active converter to produce approximately 90.7 Mg (100 tons) of blister copper per load. Converter process feed rates consist of about 650 Mg (718 tons) matte/day, 45.4 Mg (50 tons) dope/day, and 72.6 Mg (80 tons) flux/day. The finished blister copper is then poured into ladles and transported by overhead crane to one of two anode furnaces. The blister copper is completely oxidized with air and then reduced with propane or natural gas. Finished anode copper is then poured into anode molds on a single casting wheel. The anodes are cooled and subsequently loaded onto rail cars for shipment to a refinery. Anode production is about 226 Mg/day (250 tons/day) of 97.94 percent copper.

As noted previously, the calcine-laden roaster offgases pass through four parallel sets of primary and secondary cyclones. An estimated 95 percent of the dust (calcine) is recovered, and the gas stream is cooled from 565°C (1,050°F) to about 316°C (600°F). The cyclone exhaust, which has a dust loading of about 57.2 g/$Nm^3$ (25 gr/scf), then enters a venturi-type scrubber where most of the particulate is collected. The scrubbing liquid consists of weak acid which is injected into the venturi throat at a rate of 1,500 liters/min (395 gpm). The resultant pressure drop across the throat is about 406 mm (16 in.) of water. The gas stream then enters the smaller of two Peabody scrubbing towers. The larger tower is used for scrubbing and cooling the converter offgas stream. Both towers consist of a lower humidifying section and an upper cooling section. The discharge from the venturi enters the humidifying section and passes upward through a weak acid spray from spray nozzles located at the top of the section. The coarser solids are removed and the heat of the gas evaporates the weak acid water, effectively cooling the gas stream to about 68°C (155°F). The saturated gas stream then enters the cooling section where it passes through three perforated plates (four on the converter tower) for flow distribution and acid bubble formation. The weak acid flowing across the plates cools the gas stream to about 46°C (115°F). The pressure drop across both the venturi and Peabody scrubbers (which services the roaster offgases) is about 610 mm (24 in.) of water. The clean roaster gas stream (which contains about 12 percent $SO_2$) is then combined with the cleaned converter gas stream prior to entering the acid plant.

The reverberatory furnace offgases average approximately 3,682 $Nm^3$/min (130,000 scfm). They then pass through a pair of waste heat boilers at approximately 1,260°C (2,300°F) and exit at 343°C (650°F). The recovered heat is used to produce steam. The gases are then transported through a balloon flue to an ESP for particulate control. The ESP is a Koppers design and consists of four parallel chambers. The chambers have three fields in a series and a total collection area of 5,016 $m^2$ (54,000 $ft^2$). The gas retention time within the ESP is about 14 seconds. The average gas velocity is 0.5 m (1.6 ft) per second. The gases exit the precipitator at about 288°C (550°F) and are subsequently discharged through the main stack.

The offgases generated by converter blowing total 1,982 $Nm^3$/min (70,000 scfm). They are collected in water-cooled hoods and then exhausted through a gas cooler in which the gas stream is treated by a concurrently flowing, ultrasonically dispersed water spray. The cooled gas stream (371°C) flows through an induced fan plenum and into an electrostatic precipitator for particulate removal. The precipitator, manufactured by Western Precipitator, has two chambers with three fields per chamber. The total collection area of the chamber is 3,716 $m^2$ (40,000 $ft^2$). Gas retention time is about 9 sec. The average gas velocity is about 1.2 m/sec (4 ft/sec). Following the ESP, the gas stream (which contains about 5 percent $SO_2$) enters the larger of the two Peabody scrubbing towers and is treated similarly to the roaster gas stream.

After the cleaned and cooled roaster and converter gases are combined, the resultant gas stream enters three parallel trains of two mist precipitators in series, where acid mist and any remaining solids are precipitated. The gas stream (which typically contains 5 to 8 percent $SO_2$) then enters the double absorption acid plant where it is dried, the $SO_2$ converted to $SO_3$, and the $SO_3$ absorbed in acid to form strong acid. Although designed to produce 1,769 Mg (1,950 tons) of sulfuric acid per day, only about 771 Mg (850 tons) per day of 93.5 percent strength sulfuric acid is actually produced. This represents about 99.5 percent conversion.

The gas stream exiting the mist precipitator enters a drying tower where 93 percent acid is used to remove water vapor prior to entering the converter and absorbing systems. The gas stream then goes to the main blowers. One, two, or three 1,490 kW (2,000 hp) blowers are used depending on the volume of gas available for processing. The gas stream exits the main blowers into the converter system; the $SO_2$ contained in the gas stream is then converted to $SO_3$. The converter contains four layers of vanadium catalyst arranged in three passes. The first pass consists of two layers; the second and third passes each have one layer. Tube and shell heat exchangers are used to preheat the $SO_2$ gas stream to the operating temperature by utilizing the heat generated from the exothermic chemical reaction within the converter. The preheated gas enters the top of the converter and passes through the catalyst layers, exiting the converter after each pass, and entering a heat exchanger for cooling. During plant startups or during periods of low $SO_2$ gas strength, a preheater is used to raise or maintain the catalyst temperature at a level at which conversion will take place.

Two absorbing towers are used, an interpass absorber and a final absorber. The gas leaving the second converter pass goes through two heat exchangers and then to the interpass absorber where the $SO_3$ is absorbed by 98 to 99 percent acid. The gas stream then enters the final converter pass where nearly all the remaining $SO_2$ is converted.

The gas stream then enters the final absorber where the last traces of $SO_3$ are absorbed. The acid product is pumped through a series of cooling coils and then stored in any of four 5,000-ton storage tanks. The exit gas from the final absorber passes through a mist eliminator and is then exhausted through a 30.5 m (100 ft) stack. The $SO_2$ concentration in the exhaust gas is generally about 230 ppm.

Arsenic emission measurements were conducted by EPA on December 10-13, 1976. Concurrent inlet and outlet measurements were performed across the venturi and Peabody scrubbers. The scrubbers treat the roaster offgases for particulates before they are combined with converter process gases and subsequently treated for $SO_2$ in the double-contact acid plant. Additional arsenic measurements were made at the acid plant outlet.

Process conditions were carefully observed, and testing was conducted only when the subject process facilities were operating within normal operating limits. The test results are summarized in Tables C-63, C-64, and C-65.

## C.9 TAMANO SMELTER (HIBI KYODO SMELTING CO.,) JAPAN

The Tamano smelter, a toll smelting facility, is located 7 km (11 miles) southwest of Uno port and has a production capacity of 8,500 tons per month of electrolytic copper. The smelter consists of one flash furnace, three converters, two refining furnaces, and one concentrate dryer. Of the three converters, usually one is in operation, one is kept hot, and one is kept on standby.

Converter primary offgases which range between 65,000 and 75,000 $Nm^3/h$ (38,000 to 44,000 scfm) and flash furnace offgases which range between 65,000 and 80,000 $Nm^3/h$ (38,000 to 47,000 scfm) are treated in two separate pairs of electrostatic precipitators for particulate removal, then introduced into a 156,000 $Nm^3/h$ (92,000 scfm) capacity acid plant for $SO_2$ removal. $SO_2$ content of the converter and flash furnace gases usually range between 6.5 and 7.5 percent. The acid plant $SO_2$ removal efficiency is 99.7 percent. Outlet gases which contain 140 ppm $SO_2$ are vented through the main stack. A 98 percent sulfuric acid is produced at a rate of 30,000 tons/month at full capacity.

Offgases from refining furnaces are combined with gas from the power plant and passed through the concentrate dryer. Total gases from the dryer are treated in a pair of electrostatic precipitators prior to introduction to a desulfurization plant.

Each converter system is equipped with a secondary hood system for fugitive gas capture which encloses the converter mouth and ladle used for handling molten materials. The hood has two automatic front doors which are operated pneumatically. The hood has a movable roof which is slightly inclined toward the front. During closing, the roof slides to its right. During the converter operation when the hood roof is opened, fugitive emissions from the roof are controlled by an air curtain system [rated at 70,000 $Nm^3$/hr (41,000 scfm)].

The bulk of fugitive gases up to 190,000 $Nm^3$/hr (112,000 scfm) with low $SO_2$ content collected in the secondary hood are passed through a dust chamber, a baghouse system, and the main stack to the atmosphere.

Another smaller volume, fugitive gas stream with a high $SO_2$ content is continuously fed to a 200,000 $Nm^3$/hr (118,000 scfm) capacity desulfurization plant. Gas volume to the desulfurization plant includes about 72,000 $Nm^3$/hr (42,000 acfm) from the refining furnace, power plant, and flash dryer, and about 30,000 $Nm^3$/hr (18,000 scfm) leakage gases from the converters and refining furnaces.

On March 12 and 13, 1980, visible emission observations were made of the converter secondary hood system during various modes of converter operation. Tables C-75 through C-80 present the results of these observations.

## C.10 TEST DATA (TABLES)

This section contains summary data tables of the arsenic and particulate emission tests, and the visible emissions observations, conducted by EPA between December 1976, and June 1980. The following notes apply to Tables C-3 through C-65:

a) Data not reported.

b) Run no. 1 of Reverberatory-North data was performed on 6/28/77.

c) Not applicable.

## Table C-1. SUMMARY OF EMISSION TESTS

| Plant | Process Facility | Control Equipment | Sampling Location(s) | Sample Type |
|---|---|---|---|---|
| Phelps Dodge Ajo, Arizona | Acid plant from converters | Hot ESP + mist ESP + coolers | Inlet and outlet | Arsenic |
| | Reverberatory furnace (matte and slag tapping) | None | Before entering main stack | Arsenic<br>Particulate<br>$SO_2$ |
| | Converters (blow cycle) | None | Before entering main stack | Arsenic<br>Particulate<br>$SO_2$ |
| ASARCO El Paso, Texas | Multihearth roasters and reverberatory furnace | Spray chamber/ cold ESP | Inlet and outlet | Arsenic<br>Particulate |
| | Multihearth roaster discharge | Discharged into above system | Inlet | Arsenic<br>Particulate<br>$SO_2$ |
| | Reverberatory furnace (matte tapping) | Baghouse | Inlet | Arsenic<br>Particulate<br>$SO_2$ |
| | Converter building | Baghouse | Inlet and outlet | Arsenic<br>Particulate<br>$SO_2$ |
| Phelps Dodge Playas, New Mexico | Converter secondary hoods | None | Inlet | Arsenic |
| Phelps Dodge Douglas, Arizona | Multihearth roaster discharge | Baghouse | Inlet and outlet | Arsenic<br>Particulate<br>$SO_2$ |

Table C-1. SUMMARY OF EMISSION TESTS (Concluded)

| Plant | Process Facility | Control Equipment | Sampling Location(s) | Sample Type |
|---|---|---|---|---|
| ASARCO<br>Tacoma, Washington | Arsenic roasters | Baghouse | Inlet and outlet | Arsenic<br>$SO_2$ |
| | Reverberatory furnace | Cold ESP | Outlet | Arsenic |
| | Matte tappping | Cold ESP | Inlet | Arsenic |
| | Slag tapping | Cold ESP | Inlet | Arsenic |
| | Converter slag return | Cold ESP | Inlet | Arsenic |
| | Slag dumping | None | Dump site (grab samples of slag) | Arsenic |
| | Metallic arsenic | Baghouse | Inlet and outlet | Arsenic |
| | Anode furnace | None | Inlet | Arsenic (tests by ASARCo) |
| Kennecott<br>Magna, Utah | Converters | Acid Plant | Inlet | Arsenic |
| | Converter | | | |
| | Secondary hood | None | Uncontrolled duct | Arsenic<br>$SO_2$ |
| | Roll-out cycle | None | Uncontrolled duct | Arsenic<br>$SO_2$ |
| | Noranda furnace | | | |
| | Matte tapping | None | Uncontrolled duct | Arsenic<br>$SO_2$ |
| | Slag tapping | None | Uncontrolled duct | Arsenic<br>$SO_2$ |
| | Concentrate dryer | Scrubber | Outlet | Arsenic |
| Anaconda<br>Anaconda, Montana | Fluid-bed roaster, electric furnace, and converter | Spray chamber/ baghouse | Inlet and outlet | Arsenic<br>Particulate |
| Kennecott<br>Hayden, Arizona | Fluid-bed roaster | Venturi scrubber | Inlet and outlet | Arsenic |
| | Fluid-bed roaster and converters | Acid plant | Outlet | Arsenic |

Table C-2. INDEX TO ARSENIC AND PARTICULATE TEST DATA TABLES BY PROCESS FACILITY AND SAMPLE TYPE

| Process Facility | Plant | Control Equipment | Sample Type | Table Numbers in Appendix C |
|---|---|---|---|---|
| Arsenic roasters | ASARCO-Tacoma | Baghouse | Arsenic | C-3, C-4 |
| Fluid-bed roaster | Kennecott-Hayden | Venturi scrubber | Arsenic | C-63, C-64 |
| Multi-hearth roasters and reverberatory furnace | ASARCO-El Paso | Spray chamber/ cold ESP | Arsenic<br>Particulate | C-13, C-14, C-15, C-16, C-17<br>C-18, C-19, C-20, C-21, C-22, C-29 |
| Fluid-bed roaster, electric furnace, and converter | Anaconda-Anaconda | Spray chamber/ baghouse | Arsenic<br>Particulate | C-34, C-35, C-36, C-37<br>C-38, C-39, C-40, C-41 |
| Fluid-bed roaster and converters | Kennecott-Hayden | Acid plant | Arsenic | C-65 |
| Calcine discharge | ASARCO-Tacoma | None | Arsenic | C-9 |
| Roaster calcining fugitives | ASARCO-El Paso | Spray chamber/ cold ESP | Arsenic<br>Particulate | C-30<br>C-31 |
| Multi-hearth roaster fugitive discharge | Phelps Dodge-Douglas | Baghouse | Arsenic<br>Particulate | C-53, C-54<br>C-55, C-56 |
| Reverberatory furnace | ASARCO-Tacoma | Cold ESP | Arsenic | C-8 |
| Reverberatory furnace<br>Matte tapping | ASARCO-Tacoma | Cold ESP | Arsenic | C-10 |
| Matte tapping | ASARCO-El Paso | Baghouse | Arsenic<br>Particulate | C-32<br>C-33 |
| Matte and slag tapping | Phelps Dodge-Ajo | None | Arsenic<br>Particulate | C-48<br>C-49 |
| Noranda furnace<br>Matte tapping | Kennecott-Magna | None | Arsenic | C-59 |
| Reverberatory furnace<br>Slag tapping | ASARCO-Tacoma | Cold ESP | Arsenic | C-11 |
| Noranda furnace<br>Slag tapping | Kennecott-Magna | None | Arsenic | C-60 |
| Reverberatory furnace<br>Converter slag return | ASARCO-Tacoma | Cold ESP | Arsenic | C-12 |

Table C-2. INDEX TO ARSENIC AND PARTICULATE TEST DATA TABLES BY PROCESS FACILITY AND SAMPLE TYPE (Concluded)

| Process Facility | Plant | Control Equipment | Sample Type | Table Numbers in Appendix C |
|---|---|---|---|---|
| Converters | ASARCO-El Paso | Acid plant | Arsenic | C-23, C-24 |
| Converters (blow cycle) | Phelps Dodge-Ajo | None | Arsenic<br>Particulate | C-50<br>C-51 |
| Converters | Kennecott-Magna | Acid plant | Arsenic | C-58 |
| Converter fugitives | | | | |
| Full cycle | Kennecott-Magna | None | Arsenic | C-61 |
| Rollout phase | Kennecott-Magna | None | Arsenic | C-62 |
| Acid plant from converters | Phelps Dodge-Ajo | Hot ESP, mist ESP and coolers | Arsenic | C-42, C-43, C-44, C-45, C-46, C-47 |
| Converter building | ASARCO-El Paso | Baghouse | Arsenic<br>Particulate | C-25, C-26<br>C-27, C-28 |
| Converter secondary hoods | Phelps Dodge-Hidalgo | None | Arsenic | C-52 |
| Metallic arsenic process | ASARCO-Tacoma | Baghouse | Arsenic | C-5, C-6, C-7 |
| Concentrate dryers | Kennecott-Magna | Scrubber | Arsenic | C-57 |

## Table C-3. SUMMARY OF ARSENIC TEST DATA -- ROASTER BAGHOUSE INLET, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/15/78 | 9/15/78 | 9/16/78 | |
| Test Duration - min. | 80 | 84 | 80 | 81 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 173,621 | 175,277 | 184,141 | 177,680 |
| Temperature (°F) | 201 | 185 | 203 | 197 |
| Stream (vol. %): | | | | |
| Water | 4.4 | 3.4 | 4.5 | 4.1 |
| $CO_2$ | 0.2 | 0.2 | 0.2 | 0.2 |
| $O_2$ | 20.4 | 20.4 | 20.4 | 20.4 |
| $SO_2$ | 0.65 | 0.96 | 0.81 | 0.81 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.1371 | 0.1280 | 0.1101 | 0.1248 |
| gr/acf | 0.1301 | 0.1249 | 0.1092 | 0.1212 |
| lb/hr | 204.0 | 192.2 | 173.6 | 189.6 |
| Total catch | | | | |
| gr/dscf | 0.1377 | 0.1295 | 0.1111 | 0.1258 |
| gr/acf | 0.1303 | 0.1261 | 0.1103 | 0.1220 |
| lb/hr | 204.3 | 194.1 | 175.2 | 190.9 |
| Percent Isokinetic | 94.6 | 108.2 | 95.0 | |

## Table C-4. SUMMARY OF ARSENIC TEST DATA -- ROASTER BAGHOUSE OUTLET, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/15/78 | 9/15/78 | 9/16/78 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 171,887 | 174,633 | 178,671 | 175,064 |
| Temperature (°F) | 191 | 189 | 180 | 187 |
| Stream (vol. %): | | | | |
| Water | 5.9 | 5.0 | 5.6 | 5.5 |
| $CO_2$ | 0.2 | 0.2 | 0.2 | 0.2 |
| $O_2$ | 20.4 | 20.4 | 20.4 | 20.4 |
| $SO_2$ | 0.70 | 0.80 | 0.52 | 0.67 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00027 | 0.00027 | 0.00028 | 0.00027 |
| gr/acf | 0.00026 | 0.00026 | 0.00028 | 0.00026 |
| lb/hr | 0.396 | 0.401 | 0.439 | 0.412 |
| Total catch | | | | |
| gr/dscf | 0.00028 | 0.00028 | 0.00065 | 0.00040 |
| gr/acf | 0.00027 | 0.00027 | 0.00064 | 0.00039 |
| lb/hr | 0.416 | 0.428 | 0.993 | 0.612 |
| Percent Isokinetic | 99.5 | 100.1 | 102.3 | |

### Table C-5. SUMMARY OF ARSENIC TEST DATA -- ARSENIC KITCHEN BAGHOUSE INLET, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/24/78 | 9/24/78 | 9/25/78 | |
| Test Duration - min. | 96 | 96 | 96 | 96 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 14,505 | 16,590 | 17,560 | 16,218 |
| Temperature (°F) | 133 | 136 | 140 | 136 |
| Stream (vol. %): | | | | |
| Water | 3.6 | 4.0 | 2.9 | 3.5 |
| $CO_2$ | 0.2 | 0.2 | 0.2 | 0.2 |
| $O_2$ | 20.8 | 20.8 | 20.8 | 20.8 |
| $SO_2$ | 0.32 | 0.57 | 0.49 | 0.46 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.7503 | 0.6632 | 0.6593 | 0.6909 |
| gr/acf | 0.7484 | 0.6400 | 0.6400 | 0.6755 |
| lb/hr | 93.23 | 94.21 | 99.18 | 95.54 |
| Total catch | | | | |
| gr/dscf | 0.7504 | 0.6632 | 0.6695 | 0.6944 |
| gr/acf | 0.7486 | 0.6454 | 0.6402 | 0.6756 |
| lb/hr | 93.23 | 94.21 | 99.18 | 95.54 |
| Percent Isokinetic | 99.2 | 97.5 | 94.8 | |

## Table C-6. SUMMARY OF ARSENIC TEST DATA -- METALLIC ARSENIC BAGHOUSE INLET, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2* | 3 | Average |
|---|---|---|---|---|
| Date | 9/24/78 | 9/24/78 | 9/25/78 | |
| Test Duration - min. | 94 | 96 | 96 | 95 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 12,989 | 15,147 | 15,469 | 14,533 |
| Temperature (°F) | 242 | 233 | 207 | 227 |
| Stream (vol. %): | | | | |
| Water | 2.0 | 3.1 | 2.7 | 2.6 |
| $CO_2$ | 0.1 | 0.1 | 0.1 | 0.1 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.24 | 0.17 | 0.22 | 0.21 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.8440 | 0.0004 | 0.9289 | 0.5910 |
| gr/acf | 0.8343 | 0.0004 | 0.8914 | 0.5273 |
| lb/hr | 93.79 | 0.0544 | 123.0 | 72.27 |
| Total catch | | | | |
| gr/dscf | 0.8441 | 0.0006 | 0.9289 | 0.5912 |
| gr/acf | 0.8345 | 0.0006 | 0.8915 | 0.5275 |
| lb/hr | 93.81 | 0.0813 | 123.0 | 72.29 |
| Percent Isokinetic | 100.7 | 94.5 | 98.5 | |

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*During this sample run the metallic arsenic process may not have been operating.

## Table C-7. SUMMARY OF ARSENIC TEST DATA -- ARSENIC BAGHOUSE OUTLET (METALLIC AND KITCHEN), ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/24/78 | 9/24/78 | 9/25/78 | |
| Test Duration - min. | 96 | 96 | 96 | 96 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 29,109 | 35,958 | 33,726 | 32,931 |
| Temperature (°F) | 182 | 163 | 160 | 168 |
| Stream (vol. %): | | | | |
| Water | 1.0 | 2.1 | 2.8 | 2.0 |
| $CO_2$ | 0.2 | 0.2 | 0.2 | 0.2 |
| $O_2$ | 20.4 | 20.4 | 20.4 | 20.4 |
| $SO_2$ | 0.28 | 0.19 | 0.63 | 0.37 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0303 | 0.0068 | 0.0417 | 0.0253 |
| gr/acf | 0.0303 | 0.0066 | 0.0401 | 0.0248 |
| lb/hr | 7.59 | 2.11 | 12.07 | 7.26 |
| Total catch | | | | |
| gr/dscf | 0.0304 | 0.0069 | 0.0420 | 0.0255 |
| gr/acf | 0.0304 | 0.0067 | 0.0403 | 0.0249 |
| lb/hr | 7.61 | 2.13 | 12.14 | 7.29 |
| Percent Isokinetic | 97.5 | 101.3 | 97.8 | |

## Table C-8. SUMMARY OF ARSENIC TEST DATA -- REVERB ESP OUTLET, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/15/78 | 9/16/78 | 9/18/78 | |
| Test Duration - min. | 116 | 108 | 108 | 111 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 441,557 | 454,539 | 443,619 | 443,238 |
| Temperature (°F) | 220 | 214 | 188 | 207 |
| Stream (vol. %): | | | | |
| Water | 4.4 | 5.1 | 3.7 | 4.4 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.54 | 1.17 | 0.32 | 0.68 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.01632 | 0.00897 | 0.00375 | 0.00968 |
| gr/acf | 0.01538 | 0.00863 | 0.00364 | 0.00921 |
| lb/hr | 61.68 | 35.04 | 13.99 | 36.90 |
| Total catch | | | | |
| gr/dscf | 0.01669 | 0.00915 | 0.00418 | 0.01000 |
| gr/acf | 0.01574 | 0.00881 | 0.00405 | 0.00953 |
| lb/hr | 63.09 | 25.77 | 15.59 | 38.15 |
| Percent Isokinetic | 104.2 | 107.7 | 102.2 | |

## Table C-9. SUMMARY OF ARSENIC TEST DATA -- CALCINE DISCHARGE, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/20/78 | 9/20/78 | 9/21/78 | |
| Test Duration - min. | 15 | 13 | 7 | 12 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 1,239 | 1,293 | 1,524 | 1,352 |
| Temperature (°F) | 78 | 78 | 80 | 79 |
| Stream (vol. %): | | | | |
| Water | 2.5 | 0.0 | 0.0 | 0.83 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.07 | 1.32 | 0.22 | 0.54 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0939 | 0.1463 | 0.2074 | 0.1399 |
| gr/acf | 0.0936 | 0.1458 | 0.2053 | 0.1384 |
| lb/hr | 1.007 | 1.623 | 2.721 | 1.784 |
| Total catch | | | | |
| gr/dscf | 0.0946 | 0.1497 | 0.0292 | 0.1418 |
| gr/acf | 0.0943 | 0.1492 | 0.2071 | 0.1724 |
| lb/hr | 1.014 | 1.661 | 2.744 | 1.806 |
| Percent Isokinetic | 96.6 | 94.2 | 108.1 | |

## Table C-10. SUMMARY OF ARSENIC TEST DATA -- MATTE TAPPING, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/19/78 | 9/20/78 | 9/21/78 | |
| Test Duration - min. | 78 | 75 | 74 | 76 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 18,944 | 18,181 | 18,329 | 18,485 |
| Temperature (°F) | 134 | 163 | 164 | 154 |
| Stream (vol. %): | | | | |
| Water | 1.5 | 0.8 | 0.9 | 1.1 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.18 | 0.39 | 0.21 | 0.26 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.02146 | 0.10649 | 0.10332 | 0.07533 |
| gr/acf | 0.02127 | 0.10394 | 0.10130 | 0.07427 |
| lb/hr | 3.490 | 16.59 | 16.24 | 12.11 |
| Total catch | | | | |
| gr/dscf | 0.02297 | 0.10657 | 0.10344 | 0.07591 |
| gr/acf | 0.02271 | 0.10403 | 0.10142 | 0.07484 |
| lb/hr | 3.26 | 16.60 | 16.26 | 12.20 |
| Percent Isokinetic | 90.5 | 91.0 | 90.0 | |

## Table C-11. SUMMARY OF ARSENIC TEST DATA -- SLAG TAPPING, ASARCO-TACOMA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 9/19/78 | 9/20/78 | 9/20/78 | |
| Test Duration - min. | 120 | 111 | 60 | 97 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 18,351 | 16,571 | 17,219 | 17,380 |
| Temperature (°F) | 68 | 106 | 119 | 98 |
| Stream (vol. %): | | | | |
| Water | 1.5 | 2.3 | 0.3 | 1.4 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.03 | 0.05 | 0.07 | 0.05 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00340 | 0.01009 | 0.00790 | 0.00676 |
| gr/acf | 0.00335 | 0.01005 | 0.00784 | 0.00670 |
| lb/hr | 0.536 | 1.431 | 1.170 | 1.046 |
| Total catch | | | | |
| gr/dscf | 0.00348 | 0.01011 | 0.00793 | 0.00681 |
| gr/acf | 0.00343 | 0.01007 | 0.00787 | 0.00675 |
| lb/hr | 0.548 | 1.434 | 1.175 | 1.052 |
| Percent Isokinetic | 91.2 | 95.9 | 92.1 | |

## Table C-12. SUMMARY OF ARSENIC TEST DATA --CONVERTER SLAG RETURN, ASARCO-TACOMA SMELTER

| Run No. | 1 |
|---|---|
| Date | 9/19-21/78 |
| Test Duration - min. | 23 |
| Charge Rate - ton/hr | a |
| Arsenic Rate - lb/hr | a |
| Stack Effluent | |
| Flow rate (dscfm) | 23,207 |
| Temperature (°F) | 95 |
| Stream (vol. %): | |
| Water | 0.8 |
| $CO_2$ | 0.0 |
| $O_2$ | 20.0 |
| $SO_2$ | 0.7 |
| Emissions - Arsenic | |
| Probe, cyclone, and filter catch | |
| gr/dscf | 0.00139 |
| gr/acf | 0.00135 |
| lb/hr | 0.2763 |
| Total catch | |
| gr/dscf | 0.00149 |
| gr/acf | 0.00145 |
| lb/hr | 0.2962 |
| Percent Isokinetic | 91.4 |

## Table C-13. SUMMARY OF ARSENIC TEST DATA -- R & R ESP INLET (ROASTER), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/26/77 | 6/27/77 | 6/28/77 | |
| Test Duration - min. | 116 | 120 | 120 | 119 |
| Charge Rate - ton/hr | 41.2 | 28.9 | 42.7 | 37.6 |
| Arsenic Rate - lb/hr | 98.0 | 172.9 | 395.5 | 222.1 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 140,927 | 149,764 | 56,040 | 115,577 |
| Temperature (°F) | 173 | 211 | 231 | 205 |
| Stream (vol. %): | | | | |
| Water | 6.7 | 5.3 | 9.4 | 7.2 |
| $CO_2$ | 0.0 | 0.5 | 0.4 | 0.3 |
| $O_2$ | 16.6 | 18.7 | 20.5 | 18.6 |
| $SO_2$ | a | a | a | a |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0060 | 0.0080 | 0.0312 | 0.0109 |
| gr/acf | 0.0040 | 0.0052 | 0.0188 | 0.0069 |
| lb/hr | 7.200 | 10.32 | 14.99 | 9.807 |
| Total catch | | | | |
| gr/dscf | 0.0100 | 0.0103 | 0.0380 | 0.0147 |
| gr/acf | 0.0068 | 0.0067 | 0.0229 | 0.0094 |
| lb/hr | 12.12 | 13.21 | 18.25 | 13.58 |
| Percent Isokinetic | 102.4 | 66.8 | 109.2 | |

Table C-14. SUMMARY OF ARSENIC TEST DATA --
R & R ESP INLET (REVERB-NORTH), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/26/77 | 6/27/77 | 6/28/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 41.2 | 28.9 | 42.7 | 37.6 |
| Arsenic Rate - lb/hr | 98.0 | 172.9 | 395.5 | 222.1 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 38,664 | 42,288 | 40,421 | 40,458 |
| Temperature (°F) | 932 | 786 | 753 | 824 |
| Stream (vol. %): | | | | |
| Water | 6.9 | 17.9 | 19.0 | 14.6 |
| $CO_2$ | 7.6 | 7.6 | 7.5 | 7.6 |
| $O_2$ | 9.9 | 9.9 | 9.9 | 9.9 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0471 | 0.2888 | 0.2903 | 0.2123 |
| gr/acf | 0.0145 | 0.0870 | 0.0890 | 0.0646 |
| lb/hr | 15.61 | 104.7 | 100.6 | 74.95 |
| Total catch | | | | |
| gr/dscf | 0.1219 | 0.2951 | 0.3177 | 0.2475 |
| gr/acf | 0.0374 | 0.0889 | 0.0974 | 0.0753 |
| lb/hr | 40.39 | 107.0 | 110.1 | 86.81 |
| Percent Isokinetic | 103.8 | 107.1 | 112.7 | |

Table C-15. SUMMARY OF ARSENIC TEST DATA --
R & R ESP INLET (REVERB-SOUTH), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/26/77 | 6/27/77 | 6/28/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 41.2 | 28.9 | 42.7 | 37.6 |
| Arsenic Rate - lb/hr | 98.0 | 172.9 | 395.5 | 222.1 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 19,759 | 22,057 | 25,981 | 22,599 |
| Temperature (°F) | 787 | 766 | 614 | 722 |
| Stream (vol. %): | | | | |
| Water | 14.2 | 13.4 | 25.2 | 16.6 |
| $CO_2$ | 8.1 | 8.1 | 8.2 | 8.1 |
| $O_2$ | 9.3 | 9.3 | 9.2 | 9.3 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.1280 | 0.7878 | 0.6096 | 0.5272 |
| gr/acf | 0.0404 | 0.2544 | 0.1949 | 0.1692 |
| lb/hr | 21.68 | 148.9 | 135.7 | 106.8 |
| Total catch | | | | |
| gr/dscf | 0.1289 | 0.7967 | 0.6462 | 0.5444 |
| gr/acf | 0.0407 | 0.2572 | 0.2067 | 0.1747 |
| lb/hr | 21.83 | 150.6 | 143.9 | 110.5 |
| Percent Isokinetic | 95.8 | 97.7 | 113.4 | |

## Table C-16. SUMMARY OF ARSENIC TEST DATA -- R & R ESP INLET (TOTAL), ASARCO-EL PASO SMELTER*

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/26/77 | 6/27/77 | 6/28/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 41.2 | 28.9 | 42.7 | 37.6 |
| Arsenic Rate - lb/hr | 98.0 | 172.9 | 395.5 | 222.1 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 199,350 | 214,109 | 122,442 | 178,634 |
| Temperature (°F) | 381 | 382 | 485 | 411 |
| Stream (vol. %): | | | | |
| Water | 7.5 | 8.6 | 15.9 | 10.1 |
| $CO_2$ | 2.3 | 2.7 | 4.4 | 2.9 |
| $O_2$ | 14.6 | 3.3 | 14.6 | 15.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0261 | 0.1438 | 0.2426 | 0.1218 |
| gr/acf | 0.0096 | 0.0470 | 0.0812 | 0.0405 |
| lb/hr | 44.49 | 263.9 | 251.3 | 191.6 |
| Total catch | | | | |
| gr/dscf | 0.0435 | 0.1476 | 0.2594 | 0.1344 |
| gr/acf | 0.0161 | 0.0487 | 0.0865 | 0.0452 |
| lb/hr | 74.34 | 270.8 | 272.3 | 210.9 |
| Percent Isokinetic | c | c | c | |

*These data are derived from Tables C-13, C-14, and C-15.

## Table C-17. SUMMARY OF ARSENIC TEST DATA -- R & R ESP OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/26/77 | 6/27/77 | 6/28/77 | |
| Test Duration - min. | 153 | 153 | 153 | 153 |
| Charge Rate - ton/hr | 41.2 | 28.9 | 42.7 | 37.6 |
| Arsenic Rate - lb/hr | 98.0 | 172.9 | 395.5 | 222.1 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 201,030 | 209,891 | 222,719 | 211,213 |
| Temperature (°F) | 216 | 219 | 221 | 219 |
| Stream (vol. %): | | | | |
| Water | 6.8 | 6.1 | 1.4 | 4.8 |
| $CO_2$ | 3.5 | 3.5 | 0.0 | 2.3 |
| $O_2$ | 19.0 | 19.0 | 20.8 | 19.6 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0009 | 0.0031 | 0.0014 | 0.0018 |
| gr/acf | 0.0006 | 0.0019 | 0.0010 | 0.0012 |
| lb/hr | 1.603 | 5.503 | 2.761 | 3.302 |
| Total catch | | | | |
| gr/dscf | 0.0020 | 0.0041 | 0.0018 | 0.0026 |
| gr/acf | 0.0012 | 0.0026 | 0.0012 | 0.0017 |
| lb/hr | 3.401 | 7.400 | 3.393 | 4.723 |
| Percent Isokinetic | 100.4 | 100.4 | 95.4 | |

## Table C-18. SUMMARY OF PARTICULATE TEST DATA -- R & R ESP INLET (ROASTER), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 6/29/77 | 6/30/77 | |
| Test Duration - min. | 120 | 90 | 105 |
| Charge Rate - ton/hr | 43.5 | 47.6 | 45.5 |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 68,948 | 63,256 | 66,102 |
| Temperature (°F) | 190 | 197 | 193 |
| Stream (vol. %): | | | |
| Water | 9.2 | 10.5 | 9.8 |
| $CO_2$ | 1.8 | 1.4 | 1.6 |
| $O_2$ | 20.7 | 17.9 | 19.3 |
| $SO_2$ | a | a | a |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 2.245 | 1.954 | 2.106 |
| gr/acf | 1.439 | 1.222 | 1.335 |
| lb/hr | 1,327 | 1,059 | 1,199 |
| Total catch | | | |
| gr/dscf | 2.276 | 1.977 | 2.133 |
| gr/acf | 1.458 | 1.236 | 1.352 |
| lb/hr | 1,345 | 1,072 | 1,214 |
| Percent Isokinetic | 122.0 | 108.5 | |

## Table C-19. SUMMARY OF PARTICULATE TEST DATA -- R & R ESP INLET (REVERB-NORTH), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 6/28/77 | 6/30/77 | |
| Test Duration - min. | 120 | 120 | 120 |
| Charge Rate - ton/hr | 43.5 | 47.6 | 45.5 |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 40,711 | 39,889 | 40,300 |
| Temperature (°F) | 687 | 672 | 680 |
| Stream (vol. %): | | | |
| Water | 18.8 | 16.1 | 17.5 |
| $CO_2$ | 5.5 | 8.0 | 6.7 |
| $O_2$ | 5.0 | 13.2 | 9.1 |
| $SO_2$ | a | a | a |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 2.098 | 1.474 | 1.786 |
| gr/acf | 0.6814 | 0.5011 | 0.5913 |
| lb/hr | 732.0 | 504.0 | 619.2 |
| Total catch | | | |
| gr/dscf | 2.188 | 1.534 | 1.861 |
| gr/acf | 0.7106 | 0.5214 | 0.6170 |
| lb/hr | 763.3 | 524.4 | 643.9 |
| Percent Isokinetic | 109.8 | 111.6 | |

## Table C-20. SUMMARY OF PARTICULATE TEST DATA -- R & R ESP INLET (REVERB-SOUTH), ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 6/29/77 | 6/30/77 | |
| Test Duration - min. | 120 | 120 | 120 |
| Charge Rate - ton/hr | 43.5 | 47.6 | 45.5 |
| Arsenic Rate - lb/hr | a | a | a |
| **Stack Effluent** | | | |
| Flow rate (dscfm) | 22,504 | 25,680 | 24,092 |
| Temperature (°F) | 751 | 728 | 739 |
| Stream (vol. %): | | | |
| Water | 9.6 | 12.8 | 11.3 |
| $CO_2$ | 9.9 | 8.6 | 9.2 |
| $O_2$ | 3.4 | 8.5 | 6.1 |
| $SO_2$ | a | a | a |
| **Emissions - Particulate** | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 0.9863 | 5.3713 | 3.323 |
| gr/acf | 0.3377 | 1.8078 | 1.121 |
| lb/hr | 190.2 | 1,182 | 718.8 |
| Total catch | | | |
| gr/dscf | 1.0892 | 5.3974 | 3.385 |
| gr/acf | 0.3730 | 1.8166 | 1.142 |
| lb/hr | 210.1 | 1,188 | 731.3 |
| Percent Isokinetic | 109.7 | 112.3 | |

Table C-21. SUMMARY OF PARTICULATE TEST DATA -- B & R ESP INLET (TOTAL), ASARCO-EL PASO SMELTER*

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 6/29/77[b] | 6/30/77 | |
| Test Duration - min. | 120 | 120 | 120 |
| Charge Rate - ton/hr | 43.5 | 47.6 | 45.5 |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 132,163 | 128,825 | 130,494 |
| Temperature (°F) | 439 | 450 | 444 |
| Stream (vol. %): | | | |
| Water | 12.2 | 12.7 | 12.5 |
| $CO_2$ | 4.3 | 4.9 | 4.6 |
| $O_2$ | 12.9 | 14.6 | 13.7 |
| $SO_2$ | a | a | a |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 1.9854 | 2.1270 | 2.2319 |
| gr/acf | 1.0180 | 1.1154 | 1.0658 |
| lb/hr | 2,249 | 2,745 | 2,494 |
| Total catch | | | |
| gr/dscf | 2.0468 | 2.5217 | 2.2801 |
| gr/acf | 1.0431 | 1.1306 | 1.0862 |
| lb/hr | 2,318 | 2,784 | 2,548 |
| Percent Isokinetic | c | c | |

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*These data are derived from Tables C-18, C-19, and C-20.

## Table C-22. SUMMARY OF PARTICULATE TEST DATA -- R & R ESP OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 6/28/77 | 6/30/77 | |
| Test Duration - min. | 153 | 153 | 153 |
| Charge Rate - ton/hr | 43.5 | 47.6 | 45.5 |
| Arsenic Rate - lb/hr | a | a | a |
| **Stack Effluent** | | | |
| Flow rate (dscfm) | 215,414 | 233,278 | 224,346 |
| Temperature (°F) | 220 | 220 | 220 |
| Stream (vol. %): | | | |
| Water | 10.7 | 8.7 | 9.7 |
| $CO_2$ | 1.3 | 2.0 | 1.7 |
| $O_2$ | 19.0 | 20.0 | 19.5 |
| $SO_2$ | a | a | a |
| **Emissions - Particulate** | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 0.0489 | 0.0372 | 0.0427 |
| gr/acf | 0.0292 | 0.0228 | 0.0259 |
| lb/hr | 89.79 | 74.28 | 81.73 |
| Total catch | | | |
| gr/dscf | 0.0619 | 0.0478 | 0.0546 |
| gr/acf | 0.0372 | 0.0286 | 0.0327 |
| lb/hr | 114. 4 | 95.47 | 104.6 |
| Percent Isokinetic | 104.6 | 98.9 | |

## Table C-23. SUMMARY OF ARSENIC TEST DATA -- DC ACID PLANT INLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/21/77 | 6/22/77 | 6/23/77 | |
| Test Duration - min. | 105 | 96 | 96 | 99 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 58,353 | 55,189 | 54,842 | 56,128 |
| Temperature (°F) | 431 | 408 | 392 | 410 |
| Stream (vol. %): | | | | |
| Water | 2.1 | 5.7 | 4.9 | 4.2 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 17.6 | 17.2 | 13.9 | 16.2 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.9656 | 0.0810 | 0.1083 | 0.3964 |
| gr/acf | 0.4882 | 0.0405 | 0.0558 | 0.2006 |
| lb/hr | 482.9 | 38.32 | 50.89 | 196.5 |
| Total catch | | | | |
| gr/dscf | 1.0291 | 0.0997 | 0.1143 | 0.4265 |
| gr/acf | 0.5203 | 0.0498 | 0.0589 | 0.2158 |
| lb/hr | 514.6 | 47.16 | 53.73 | 211.3 |
| Percent Isokinetic | 98.5 | 112.2 | 106.0 | |

## Table C-24. SUMMARY OF ARSENIC TEST DATA -- DC ACID PLANT OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | 4 | Average |
|---|---|---|---|---|---|
| Date | 6/21/77 | 6/22/77 | 6/23/77 | 6/24/77 | |
| Test Duration - min. | 132 | 132 | 132 | 132 | 132 |
| Charge Rate - ton/hr | a | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a | a |
| **Stack Effluent** | | | | | |
| Flow rate (dscfm) | 68,108 | 66,574 | 66,214 | 64,643 | 65,429 |
| Temperature (°F) | 147 | 147 | 151 | 156 | 153 |
| Stream (vol. %): | | | | | |
| Water | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 17.6 | 16.0 | 13.6 | 11.7 | 14.7 |
| $SO_2$ | a | a | a | a | a |
| **Emissions - Arsenic** | | | | | |
| Probe, cyclone, and filter catch | | | | | |
| gr/dscf | 0.0001 | 0.0008 | 0.0005 | 0.0001 | 0.0004 |
| gr/acf | 0.0001 | 0.0006 | 0.0004 | 0.0001 | 0.0003 |
| lb/hr | 0.043 | 0.452 | 0.267 | 0.050 | 0.203 |
| Total catch | | | | | |
| gr/dscf | 0.0001 | 0.0014 | 0.0005 | 0.0002 | 0.0005 |
| gr/acf | 0.0001 | 0.0010 | 0.0004 | 0.0001 | 0.0004 |
| lb/hr | 0.048 | 0.783 | 0.277 | 0.085 | 0.298 |
| Percent Isokinetic | 73.0 | 76.2 | 96.4 | 97.4 | |

## Table C-25. SUMMARY OF ARSENIC TEST DATA -- CONVERTER BUILDING BAGHOUSE INLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/18/78 | 1/19/78 | 1/23/78 | |
| Test Duration - min. | 101 | 100 | 100 | 100 |
| Charge Rate - ton/hr | 41.7 | 32.7 | 37.1 | 37.2 |
| Arsenic Rate - lb/hr | 212.0 | 116.1 | 165.2 | 164.4 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 521,956 | 528,463 | 506,479 | 527,497 |
| Temperature (°F) | 100 | 97 | 123 | 107 |
| Stream (vol. %): | | | | |
| Water | 1.0 | 0.4 | 1.1 | 0.8 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00270 | 0.00090 | 0.00067 | 0.00142 |
| gr/acf | 0.00240 | 0.00082 | 0.00058 | 0.00123 |
| lb/hr | 12.11 | 4.28 | 2.88 | 6.42 |
| Total catch | | | | |
| gr/dscf | 0.00272 | 0.00091 | 0.00067 | 0.00143 |
| gr/acf | 0.00242 | 0.00083 | 0.00058 | 0.00128 |
| lb/hr | 12.12 | 4.33 | 2.90 | 6.45 |
| Percent Isokinetic | 95.0 | 94.2 | 93.8 | |

Table C-26. SUMMARY OF ARSENIC TEST DATA -- CONVERTER BUILDING BAGHOUSE OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/18/78 | 1/19/78 | 1/23/78 | |
| Test Duration - min. | 160 | 160 | 200 | 173 |
| Charge Rate - ton/hr | 41.7 | 32.7 | 37.1 | 37.2 |
| Arsenic Rate - lb/hr | 212.0 | 116.1 | 165.2 | 164.4 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 437,609 | 526,565 | 490,455 | 491,062 |
| Temperature (°F) | 101 | 99 | 124 | 108 |
| Stream (vol. %): | | | | |
| Water | 0.0 | 0.7 | 1.3 | 0.7 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00017 | 0.000005 | 0.000006 | 0.00006 |
| gr/acf | 0.00015 | 0.000005 | 0.000005 | 0.00005 |
| lb/hr | 0.672 | 0.025 | 0.026 | 0.241 |
| Total catch | | | | |
| gr/dscf | 0.00017 | 0.000010 | 0.000007 | 0.00006 |
| gr/acf | 0.00015 | 0.000009 | 0.000006 | 0.00005 |
| lb/hr | 0.672 | 0.040 | 0.027 | 0.246 |
| Percent Isokinetic | 105.7 | 101.1 | 102.6 | |

## Table C-27. SUMMARY OF PARTICULATE TEST DATA -- CONVERTER BUILDING BAGHOUSE INLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/17/78 | 1/18/78 | 1/21/78 | |
| Test Duration - min. | 105 | 100 | 100 | 102 |
| Charge Rate - ton/hr | a | 41.7 | 43.8 | 42.8 |
| Arsenic Rate - lb/hr | a | a | a | a |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 435,427 | 514,279 | 510,318 | 486,675 |
| Temperature (°F) | 115 | 113 | 115 | 114 |
| Stream (vol. %): | | | | |
| Water | 1.2 | 1.4 | 1.4 | 1.3 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | 0.017 | 0.001 | 0.001 | 0.006 |
| **Emissions - Particulate** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0263 | 0.0080 | 0.0309 | 0.0215 |
| gr/acf | 0.0235 | 0.0202 | 0.0276 | 0.0238 |
| lb/hr | 98.25 | 101.9 | 134.5 | 111.5 |
| Total catch | | | | |
| gr/dscf | 0.2812 | 0.2686 | 0.031 | 0.2749 |
| gr/acf | 0.2507 | 0.2346 | 0.027 | 0.2427 |
| lb/hr | 1,049 | 1,181 | 134.5 | 788.0 |
| Percent Isokinetic | 109.7 | 94.5 | 96.3 | |

Table C-28. SUMMARY OF PARTICULATE TEST DATA -- CONVERTER BUILDING BAGHOUSE OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/17/78 | 1/18/78 | 1/21/78 | |
| Test Duration - min. | 120 | 160 | 160 | 146 |
| Charge Rate - ton/hr | a | 41.7 | 43.8 | 42.8 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 526,089 | 471,191 | 496,907 | 498,062 |
| Temperature (°F) | 114 | 118 | 114 | 115 |
| Stream (vol. %): | | | | |
| Water | 1.0 | 0.9 | 1.1 | 1.0 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0051 | 0.0011 | 0.0005 | 0.0072 |
| gr/acf | 0.0045 | 0.0010 | 0.0004 | 0.0018 |
| lb/hr | 22.87 | 4.45 | 1.96 | 9.76 |
| Total catch | | | | |
| gr/dscf | 0.1117 | 0.1402 | 0.0081 | 0.0867 |
| gr/acf | 0.0979 | 0.1260 | 0.0072 | 0.0770 |
| lb/hr | 503.8 | 567.4 | 34.29 | 368.5 |
| Percent Isokinetic | 98.0 | 97.7 | 103.7 | |

Table C-29. SUMMARY OF PARTICULATE TEST DATA -- ROASTER/REVERBERATORY ESP OUTLET, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/26/78 | 1/26/78 | 1/27/78 | |
| Test Duration - min. | 144 | 147 | 150 | 147 |
| Charge Rate - ton/hr | 64.0 | 64.0 | 60.1 | 62.7 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 200,352 | 207,295 | 202,651 | 203,433 |
| Temperature (°F) | 219 | 205 | 220 | 215 |
| Stream (vol. %): | | | | |
| Water | 6.0 | 6.2 | 7.3 | 6.5 |
| $CO_2$ | 3.0 | 3.0 | 3.0 | 3.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0608 | 0.0909 | 0.0411 | 0.0643 |
| gr/acf | 0.0520 | 0.0787 | 0.0365 | 0.0577 |
| lb/hr | 104.0 | 160.7 | 71.1 | 112.0 |
| Total catch | | | | |
| gr/dscf | 0.1054 | 0.1118 | 0.0553 | 0.0909 |
| gr/acf | 0.0901 | 0.0968 | 0.0491 | 0.0788 |
| lb/hr | 180.2 | 197.7 | 95.7 | 157.9 |
| Percent Isokinetic | 94.2 | 90.6 | 98.8 | |

Table C-30. SUMMARY OF ARSENIC TEST DATA -- CALCINE DISCHARGE DUCT, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/24/78 | 1/24/78 | 1/24/78 | |
| Test Duration - min. | 60 | 60 | 60 | 60 |
| Charge Rate - ton/hr | 21.3 | 21.3 | 21.3 | 21.3 |
| Arsenic Rate - lb/hr | 83.9 | 83.9 | 83.9 | 83.9 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 7,705 | 7,809 | 7,659 | 7,724 |
| Temperature (°F) | 56 | 57 | 61 | 58 |
| Stream (vol. %): | | | | |
| Water | 0.1 | 0.4 | 0.3 | 0.3 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0049 | 0.0014 | 0.0034 | 0.0032 |
| gr/acf | 0.0045 | 0.0012 | 0.0030 | 0.0029 |
| lb/hr | 0.326 | 0.092 | 0.223 | 0.214 |
| Total catch | | | | |
| gr/dscf | 0.0050 | 0.0014 | 0.0034 | 0.0033 |
| gr/acf | 0.0045 | 0.0018 | 0.0031 | 0.0031 |
| lb/hr | 0.332 | 0.094 | 0.224 | 0.217 |
| Percent Isokinetic | 93.4 | 100.7 | 94.0 | |

## Table C-31. SUMMARY OF PARTICULATE TEST DATA -- CALCINE DISCHARGE DUCT, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/24/78 | 1/25/78 | 1/25/78 | |
| Test Duration - min. | 57 | 60 | 60 | 59 |
| Charge Rate - ton/hr | 23.5 | 37.8 | 37.8 | 33.0 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 7,932 | 7,737 | 7,394 | 7,687 |
| Temperature (°F) | 57 | 88 | 96 | 80 |
| Stream (vol. %): | | | | |
| Water | 0.4 | 0.3 | 1.0 | 0.6 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | 0.014 | 0.005 | 0.001 | 0.007 |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0512 | 0.0338 | 0.0090 | 0.0313 |
| gr/acf | 0.0457 | 0.0305 | 0.0080 | 0.0281 |
| lb/hr | 3.49 | 2.23 | 0.57 | 2.09 |
| Total catch | | | | |
| gr/dscf | 0.1861 | 0.1481 | 0.3196 | 0.2179 |
| gr/acf | 0.1630 | 0.1336 | 0.2861 | 0.1620 |
| lb/hr | 12.70 | 10.11 | 20.23 | 14.34 |
| Percent Isokinetic | 96.1 | 103.6 | 100.6 | |

## Table C-32. SUMMARY OF ARSENIC TEST DATA -- MATTE TAPPING DUCT, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/20/78 | 1/20/78 | 1/25/78 | |
| Test Duration - min. | 240 | 240 | 396 | 292 |
| Charge Rate - ton/hr | 69.2 | 69.2 | 39.1 | 59.1 |
| Arsenic Rate - lb/hr | 474.2 | 474.2 | 176.0 | 374.8 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 23,296 | 26,367 | 27,418 | 25,694 |
| Temperature (°F) | 105 | 98 | 73 | 92 |
| Stream (vol. %): | | | | |
| Water | 0.3 | 0.0 | 0.0 | 0.1 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0029 | 0.0026 | 0.0012 | 0.0022 |
| gr/acf | 0.0024 | 0.0022 | 0.0010 | 0.0019 |
| lb/hr | 0.578 | 0.598 | 0.288 | 0.488 |
| Total catch | | | | |
| gr/dscf | 0.0029 | 0.0027 | 0.0012 | 0.00003 |
| gr/acf | 0.0025 | 0.0022 | 0.0010 | 0.0019 |
| lb/hr | 0.580 | 0.600 | 0.288 | 0.489 |
| Percent Isokinetic | 94.5 | 90.2 | 86.9 | |

## Table C-33. SUMMARY OF PARTICULATE TEST DATA -- MATTE TAPPING DUCT, ASARCO-EL PASO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 1/25/78 | 1/26/78 | 1/26/78 | |
| Test Duration - min. | 389 | 360 | 360 | 369 |
| Charge Rate - ton/hr | 39.1 | 67.6 | 67.6 | 58.1 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 26,871 | 27,370 | 26,802 | 27,015 |
| Temperature (°F) | 73 | 82 | 82 | 79 |
| Stream (vol. %): | | | | |
| Water | 0.8 | 0.0 | 0.0 | 0.3 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.5 | 20.5 | 20.5 | 20.5 |
| $SO_2$ | 0.006 | 0.009 | 0.020 | 0.012 |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0055 | 0.0193 | 0.0164 | 0.0134 |
| gr/acf | 0.0047 | 0.0161 | 0.0100 | 0.0103 |
| lb/hr | 1.07 | 3.77 | 3.09 | 2.64 |
| Total catch | | | | |
| gr/dscf | 0.0978 | 0.1632 | 0.0712 | 0.1441 |
| gr/acf | 0.0836 | 0.1370 | 0.1366 | 0.1191 |
| lb/hr | 19.12 | 32.14 | 32.27 | 27.84 |
| Percent Isokinetic | 93.7 | 96.0 | 91.9 | |

## Table C-34. SUMMARY OF ARSENIC TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET-WEST, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/20/77 | 4/21/77 | 4/22/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 97 | 92 | 92 | 93.7 |
| Arsenic Rate - lb/hr | 2,173 | 1,490 | 1,914 | 1,857 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 76,274 | 74,002 | 77,402 | 75,893 |
| Temperature (°F) | 517 | 511 | 466 | 498 |
| Stream (vol. %): | | | | |
| Water | 11.5 | 12.2 | 9.7 | 11.1 |
| $CO_2$ | 4.0 | 4.2 | 0.2 | 2.8 |
| $O_2$ | 16.4 | 18.2 | 19.8 | 18.1 |
| $SO_2$ | a | a | a | a |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.4012 | 0.3550 | 0.3086 | 0.3547 |
| gr/acf | 0.1551 | 0.1383 | 0.1309 | 0.1414 |
| lb/hr | 262.2 | 225.1 | 204.7 | 230.6 |
| Total catch | | | | |
| gr/dscf | 0.4063 | 0.3706 | 0.3153 | 0.3638 |
| gr/acf | 0.1571 | 0.1444 | 0.1337 | 0.1450 |
| lb/hr | 265.6 | 235.0 | 209.1 | 236.4 |
| Percent Isokinetic | 92.9 | 101.9 | 100.7 | |

## Table C-35. SUMMARY OF ARSENIC TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET-EAST, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/20/77 | 4/21/77 | 4/22/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 97 | 92 | 92 | 93.7 |
| Arsenic Rate - lb/hr | 2,173 | 1,490 | 1,914 | 1,857 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 80,193 | 86,350 | 86,889 | 84,477 |
| Temperature (°F) | 535 | 523 | 475 | 511 |
| Stream (vol. %): | | | | |
| Water | 12.9 | 12.8 | 10.3 | 12.0 |
| $CO_2$ | 0.8 | 0.5 | 0.0 | 0.4 |
| $O_2$ | 17.4 | 19.5 | 19.6 | 18.8 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.4939 | 0.3771 | 0.2559 | 0.3725 |
| gr/acf | 0.1846 | 0.1442 | 0.1069 | 0.1442 |
| lb/hr | 339.4 | 279.0 | 190.6 | 267.8 |
| Total catch | | | | |
| gr/dscf | 0.4992 | 0.3864 | 0.2690 | 0.3818 |
| gr/acf | 0.1866 | 0.1477 | 0.1124 | 0.1479 |
| lb/hr | 343.1 | 285.9 | 200.3 | 274.7 |
| Percent Isokinetic | 98.7 | 96.2 | 98.5 | |

Table C-36. SUMMARY OF ARSENIC TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET (TOTAL), ANACONDA-ANACONDA SMELTER*

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/20/77 | 4/21/77 | 4/22/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 97 | 92 | 92 | 93.7 |
| Arsenic Rate - lb/hr | 2,173 | 1,490 | 1,914 | 1,857 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 156,467 | 160,352 | 164,291 | 160,370 |
| Temperature (°F) | 526 | 517 | 471 | 505 |
| Stream (vol. %): | | | | |
| Water | 12.2 | 12.5 | 10.0 | 11.6 |
| $CO_2$ | 2.4 | 2.2 | 0.1 | 1.5 |
| $O_2$ | 16.9 | 18.9 | 19.7 | 18.5 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.4588 | 0.3669 | 0.2807 | 0.3595 |
| gr/acf | 0.1702 | 0.1415 | 0.1182 | 0.1433 |
| lb/hr | 601.6 | 504.1 | 395.3 | 498.4 |
| Total catch | | | | |
| gr/dscf | 0.4539 | 0.3791 | 0.2908 | 0.3733 |
| gr/acf | 0.1722 | 0.1462 | 0.1224 | 0.1465 |
| lb/hr | 608.7 | 520.9 | 409.4 | 511.1 |
| Percent Isokinetic | c | c | c | |

*These data are derived from Tables C-34 and C-35.

### Table C-37. SUMMARY OF ARSENIC TEST DATA -- SPRAY CHAMBER/BAGHOUSE OUTLET, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/20/77 | 4/21/77 | 4/22/77 | |
| Test Duration - min. | 128 | 128 | 128 | 128 |
| Charge Rate - ton/hr | 97 | 92 | 92 | 93.7 |
| Arsenic Rate - lb/hr | 2,173 | 1,490 | 1,914 | 1,857 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 153,594 | 156,349 | 164,134 | 158,026 |
| Temperature (°F) | 210 | 215 | 214 | 213 |
| Stream (vol. %): | | | | |
| Water | 19.1 | 19.3 | 17.7 | 18.7 |
| $CO_2$ | 3.6 | 4.5 | 3.5 | 3.9 |
| $O_2$ | 17.0 | 17.5 | 16.9 | 17.1 |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0018 | 0.0023 | 0.0036 | 0.0026 |
| gr/acf | 0.0009 | 0.0012 | 0.0019 | 0.0013 |
| lb/hr | 2.41 | 3.07 | 5.00 | 3.52 |
| Total catch | | | | |
| gr/dscf | 0.0031 | 0.0041 | 0.0053 | 0.0042 |
| gr/acf | 0.0016 | 0.0021 | 0.0028 | 0.0015 |
| lb/hr | 4.02 | 5.53 | 7.45 | 5.71 |
| Percent Isokinetic | 100.4 | 102.2 | 98.7 | |

Table C-38. SUMMARY OF PARTICULATE TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET-WEST, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/25/78 | 4/25/78 | 4/26/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 86 | 86 | 70 | 80.7 |
| Arsenic Rate - lb/hr | a | a | a | a |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 77,031 | 80,363 | 75,458 | 77,617 |
| Temperature (°F) | 535 | 546 | 573 | 551 |
| Stream (vol. %): | | | | |
| Water | 13.2 | 12.3 | 11.2 | 12.2 |
| $CO_2$ | 2.3 | 1.4 | 1.0 | 1.6 |
| $O_2$ | 16.2 | 16.9 | 19.4 | 17.5 |
| $SO_2$ | a | a | a | a |
| **Emissions - Particulate** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 7.25 | 6.20 | 6.40 | 6.61 |
| gr/acf | 2.74 | 2.34 | 2.37 | 2.48 |
| lb/hr | 4,786 | 4,267 | 4,139 | 4,397 |
| Total catch | | | | |
| gr/dscf | 7.36 | 6.25 | 6.43 | 6.68 |
| gr/acf | 2.78 | 2.36 | 2.38 | 2.51 |
| lb/hr | 4,858 | 4,306 | 4,161 | 4,442 |
| Percent Isokinetic | 97.4 | 98.5 | 100.9 | |

## Table C-39. SUMMARY OF PARTICULATE TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET-EAST, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/25/77 | 4/25/77 | 4/26/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 86 | 86 | 70 | 80.7 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 85,140 | 81,352 | 85,669 | 84,054 |
| Temperature (°F) | 541 | 555 | 577 | 558 |
| Stream (vol. %): | | | | |
| Water | 11.4 | 12.2 | 13.5 | 12.4 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.1 | 19.4 | 19.6 | 19.7 |
| $SO_2$ | a | a | a | a |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 5.72 | 5.67 | 5.93 | 5.78 |
| gr/acf | 2.20 | 2.13 | 2.13 | 2.15 |
| lb/hr | 4,170 | 3,952 | 4,352 | 4,162 |
| Total catch | | | | |
| gr/dscf | 5.83 | 5.76 | 6.05 | 5.88 |
| gr/acf | 2.24 | 2.16 | 2.18 | 2.19 |
| lb/hr | 4,254 | 4,013 | 4,442 | 4,240 |
| Percent Isokinetic | 97.9 | 96.1 | 98.4 | |

## Table C-40. SUMMARY OF PARTICULATE TEST DATA -- SPRAY CHAMBER/BAGHOUSE INLET (TOTAL), ANACONDA-ANACONDA SMELTER*

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/25/77 | 4/25/77 | 4/26/77 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | 86 | 86 | 70 | 80.7 |
| Arsenic Rate - lb/hr | a | a | a | a |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 162,171 | 161,715 | 161,127 | 161,671 |
| Temperature (°F) | 538 | 551 | 575 | 555 |
| Stream (vol. %): | | | | |
| Water | 12.3 | 12.3 | 12.3 | 12.3 |
| $CO_2$ | 1.2 | 0.7 | 0.5 | 0.8 |
| $O_2$ | 18.2 | 18.2 | 19.5 | 18.7 |
| $SO_2$ | a | a | a | a |
| **Emissions - Particulate** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 6.45 | 5.93 | 6.15 | 6.18 |
| gr/acf | 2.46 | 2.23 | 2.24 | 2.31 |
| lb/hr | 8,956 | 8,219 | 8,491 | 8,559 |
| Total catch | | | | |
| gr/dscf | 6.56 | 6.00 | 6.23 | 6.26 |
| gr/acf | 2.50 | 2.26 | 2.27 | 2.34 |
| lb/hr | 9,112 | 8,319 | 8,603 | 8,682 |
| Percent Isokinetic | c | c | c | |

*These data are derived from Tables C-38 and C-39.

## Table C-41. SUMMARY OF PARTICULATE TEST DATA -- SPRAY CHAMBER/BAGHOUSE OUTLET, ANACONDA-ANACONDA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 4/25/77 | 4/25/77 | 4/26/77 | |
| Test Duration - min. | 128 | 128 | 128 | 128 |
| Charge Rate - ton/hr | 86 | 86 | 70 | 80.7 |
| Arsenic Rate - lb/hr | a | a | a | a |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 170,466 | 158,252 | 165,400 | 164,706 |
| Temperature (°F) | 217 | 218 | 213 | 216 |
| Stream (vol. %): | | | | |
| Water | 16.4 | 19.4 | 19.7 | 18.5 |
| $CO_2$ | 4.5 | 4.8 | 5.2 | 4.8 |
| $O_2$ | 18.5 | 18.2 | 17.5 | 18.1 |
| $SO_2$ | a | a | a | a |
| **Emissions - Particulate** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0220 | 0.0162 | 0.0228 | 0.0204 |
| gr/acf | 0.0119 | 0.0085 | 0.0115 | 0.0107 |
| lb/hr | 32.1 | 22.0 | 32.3 | 28.9 |
| Total catch | | | | |
| gr/dscf | 0.1387 | 0.0667 | 0.0288 | 0.0789 |
| gr/acf | 0.0754 | 0.0349 | 0.0146 | 0.0421 |
| lb/hr | 202.6 | 90.4 | 40.8 | 112.5 |
| Percent Isokinetic | 96.7 | 99.4 | 97.7 | |

## Table C-42. SUMMARY OF ARSENIC TEST DATA -- REVERBERATORY ESP INLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 7/13/76 | 7/14/76 | 7/14/76 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | 59.3 | 72.8 | 75.4 | 69.2 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 58,814 | 59,583 | 60,150 | 59,516 |
| Temperature (°F) | 622 | 602 | 639 | 621 |
| Stream (vol. %): | | | | |
| Water | 18.0 | 18.6 | 16.0 | 17.5 |
| $CO_2$ | a | a | a | a |
| $O_2$ | a | a | a | a |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.1076 | 0.1326 | 0.1394 | 0.1265 |
| gr/acf | 0.0527 | 0.0662 | 0.0673 | 0.0621 |
| lb/hr | 54.3 | 67.8 | 71.9 | 64.7 |
| Total catch | | | | |
| gr/dscf | 0.1172 | 0.1423 | 0.1459 | 0.1351 |
| gr/acf | 0.0574 | 0.0710 | 0.0704 | 0.0663 |
| lb/hr | 59.1 | 72.7 | 75.2 | 69.0 |
| Percent Isokinetic | 154 | 152 | 147 | |

## Table C-43. SUMMARY OF ARSENIC TEST DATA -- REVERBERATORY ESP OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 7/13/76 | 7/14/76 | 7/14/76 | |
| Test Duration - min. | 120 | 120 | 120 | 120 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | 53.7 | 44.8 | 51.3 | 49.9 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 68,030 | 66,275 | 68,738 | 67,681 |
| Temperature (°F) | 595 | 610 | 580 | 595 |
| Stream (vol. %): | | | | |
| Water | 15.3 | 15.4 | 13.5 | 14.7 |
| $CO_2$ | a | a | a | a |
| $O_2$ | a | a | a | a |
| $SO_2$ | a | a | a | a |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0603 | 0.0376 | 0.0302 | 0.0427 |
| gr/acf | 0.0303 | 0.0186 | 0.0154 | 0.0214 |
| lb/hr | 35.2 | 21.4 | 17.8 | 24.8 |
| Total catch | | | | |
| gr/dscf | 0.0919 | 0.0786 | 0.0868 | 0.0858 |
| gr/acf | 0.0462 | 0.0389 | 0.0442 | 0.0431 |
| lb/hr | 53.6 | 44.6 | 51.1 | 49.8 |
| Percent Isokinetic | 145 | 147 | 136 | |

Table C-44. SUMMARY OF ARSENIC TEST DATA -- CONVERTER ESP INLET NO. 1, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2* | 3 | Average |
|---|---|---|---|---|
| Date | 6/13/78 | | 6/15/78 | |
| Test Duration - min. | 144 | | 144 | 144 |
| Charge Rate - ton/hr | 17.7 | | 17.7 | 17.7 |
| Arsenic Rate - lb/hr | 38.9 | | 95.6 | 67.2 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 28,075 | | 26,638 | 27,358 |
| Temperature (°F) | 379 | | 405 | 392 |
| Stream (vol. %): | | | | |
| Water | 0.0 | | 4.0 | 2.0 |
| $CO_2$ | 0.0 | | 0.0 | 0.0 |
| $O_2$ | 20.0 | | 20.0 | 20.0 |
| $SO_2$ | 3.83 | | 4.18 | 4.01 |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.000038 | | 0.000010 | 0.000024 |
| gr/acf | 0.000032 | | 0.000008 | 0.000020 |
| lb/hr | 0.0091 | | 0.0022 | 0.0056 |
| Total catch | | | | |
| gr/dscf | 0.000100 | | 0.000049 | 0.000073 |
| gr/acf | 0.000083 | | 0.000042 | 0.000063 |
| lb/hr | 0.0241 | | 0.0112 | 0.0062 |
| Percent Isokinetic | 126.4 | | 95.2 | |

*Test run aborted.

## Table C-45. SUMMARY OF ARSENIC TEST DATA -- CONVERTER ESP INLET NO. 2, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/13/78 | 6/14/78 | 6/15/78 | |
| Test Duration - min. | 144 | 144 | 144 | 144 |
| Charge Rate - ton/hr | 17.7 | 19.7 | 17.7 | 18.3 |
| Arsenic Rate - lb/hr | 38.9 | 63.0 | 95.6 | 65.8 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 34,282 | 28,312 | 29,265 | 30,621 |
| Temperature (°F) | 389 | 358 | 404 | 384 |
| Stream (vol. %): | | | | |
| Water | 0.3 | 0.8 | 0.0 | 0.4 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 2.59 | 3.03 | 7.60 | 4.41 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.000003 | 0.000010 | 0.000005 | 0.000006 |
| gr/acf | 0.000002 | 0.000009 | 0.000004 | 0.000005 |
| lb/hr | 0.0095 | 0.0026 | 0.0011 | 0.0044 |
| Total catch | | | | |
| gr/dscf | 0.000003 | 0.000013 | 0.000013 | 0.000010 |
| gr/acf | 0.000003 | 0.000011 | 0.000011 | 0.000008 |
| lb/hr | 0.0097 | 0.0031 | 0.0029 | 0.0053 |
| Percent Isokinetic | 123.2 | 92.1 | 99.0 | |

## Table C-46. SUMMARY OF ARSENIC TEST DATA -- CONVERTER ESP OUTLET (ACID PLANT INLET), PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/13/78 | 6/14/78 | 6/15/78 | |
| Test Duration - min. | 106 | 111 | 109 | 109 |
| Charge Rate - ton/hr | 17.7 | 19.7 | 17.7 | 18.3 |
| Arsenic Rate - lb/hr | 38.9 | 63.0 | 95.6 | 65.8 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 41,016 | 39,021 | 29,692 | 36,578 |
| Temperature (°F) | 374 | 360 | 342 | 359 |
| Stream (vol. %): | | | | |
| Water | 3.0 | 2.2 | 3.7 | 3.0 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 4.98 | 2.83 | 3.99 | 3.93 |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.000033 | 0.000010 | 0.000016 | 0.000020 |
| gr/acf | 0.000028 | 0.000008 | 0.000014 | 0.000017 |
| lb/hr | 0.0117 | 0.0033 | 0.0042 | 0.0064 |
| Total catch | | | | |
| gr/dscf | 0.000035 | 0.000011 | 0.000040 | 0.000029 |
| gr/acf | 0.000030 | 0.000010 | 0.000035 | 0.000025 |
| lb/hr | 0.0123 | 0.0037 | 0.0104 | 0.0088 |
| Percent Isokinetic | 101.3 | 102.1 | 106.8 | |

## Table C-47. SUMMARY OF ARSENIC TEST DATA -- ACID PLANT OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 6/13/78 | 6/14/78 | 6/15/78 | |
| Test Duration - min. | 108 | 108 | 108 | 108 |
| Charge Rate - ton/hr | 17.7 | 19.7 | 17.7 | 18.3 |
| Arsenic Rate - lb/hr | 38.9 | 63.0 | 95.6 | 65.8 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 47,556 | 43,862 | 36,016 | 42,478 |
| Temperature (°F) | 140 | 164 | 128 | 144 |
| Stream (vol. %): | | | | |
| Water | 1.2 | 0.5 | 0.0 | 0.6 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.24 | 0.12 | 0.08 | 0.15 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0000020 | 0.0000026 | 0.0000109 | 0.0000052 |
| gr/acf | 0.0000017 | 0.0000020 | 0.0000091 | 0.0000043 |
| lb/hr | 0.0009 | 0.0011 | 0.0033 | 0.0018 |
| Total catch | | | | |
| gr/dscf | 0.0000030 | 0.0000048 | 0.0000120 | 0.0000068 |
| gr/acf | 0.0000026 | 0.0000039 | 0.0000021 | 0.0000052 |
| lb/hr | 0.0013 | 0.0015 | 0.0040 | 0.0022 |
| Percent Isokinetic | 100.6 | 97.4 | 97.0 | |

Table C-48. SUMMARY OF ARSENIC TEST DATA -- MATTE TAPPING HOOD OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/10/78 | 5/11/78 | 5/12/78 | |
| Test Duration - min. | 192 | 120 | 120 | 144 |
| Charge Rate - ton/hr | 43.9 | 40.2 | 48.1 | 44.1 |
| Arsenic Rate - lb/hr | 776.7 | 741.3 | 882.5 | 800.1 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 63,758 | 72,351 | 69,333 | 69,056 |
| Temperature (°F) | 111 | 111 | 120 | 114 |
| Stream (vol. %): | | | | |
| Water | 0.9 | 0.9 | 1.2 | 1.0 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.02 | 0.04 | 0.04 | 0.03 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00045 | 0.00076 | 0.00051 | 0.00058 |
| gr/acf | 0.00042 | 0.00070 | 0.00046 | 0.00052 |
| lb/hr | 0.248 | 0.472 | 0.300 | 0.340 |
| Total catch | | | | |
| gr/dscf | 0.00066 | 0.00082 | 0.00057 | 0.00069 |
| gr/acf | 0.00062 | 0.00075 | 0.00052 | 0.00063 |
| lb/hr | 0.365 | 0.508 | 0.344 | 0.406 |
| Percent Isokinetic | 104.2 | 95.8 | 97.2 | |

## Table C-49. SUMMARY OF PARTICULATE TEST DATA -- MATTE TAPPING OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 5/10/78 | 5/11/78 | |
| Test Duration - min. | 192 | 120 | 156 |
| Charge Rate - ton/hr | 43.9 | 40.2 | 42.0 |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 68,930 | 73,362 | 71,140 |
| Temperature (°F) | 110 | 112 | 111 |
| Stream (vol. %): | | | |
| Water | 0.7 | 0.3 | 0.5 |
| $CO_2$ | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.02 | 0.04 | 0.03 |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 0.0133 | 0.0226 | 0.0179 |
| gr/acf | 0.0123 | 0.0207 | 0.0165 |
| lb/hr | 7.89 | 14.31 | 11.10 |
| Total catch | | | |
| gr/dscf | 0.0160 | 0.0466 | 0.0313 |
| gr/acf | 0.0148 | 0.0427 | 0.0288 |
| lb/hr | 9.51 | 29.49 | 19.50 |
| Percent Isokinetic | 108.7 | 100.1 | |

## Table C-50. SUMMARY OF ARSENIC TEST DATA -- CONVERTER SECONDARY HOOD OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/10/78 | 5/11/78 | 5/12/78 | |
| Test Duration - min. | 244 | 120 | 120 | 161 |
| Charge Rate - ton/hr | 27.3 | 10.6 | 28.4 | 22.1 |
| Arsenic Rate - lb/hr | 2.9 | 1.1 | 3.0 | 2.3 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 85,659 | 87,444 | 85,957 | 86,353 |
| Temperature (°F) | 142 | 162 | 158 | 154 |
| Stream (vol. %): | | | | |
| Water | 0.5 | 1.2 | 0.6 | 0.8 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.23 | 0.34 | 0.37 | 0.31 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00247 | 0.00245 | 0.00131 | 0.00208 |
| gr/acf | 0.00224 | 0.00225 | 0.00121 | 0.00190 |
| lb/hr | 1.813 | 1.834 | 0.961 | 1.536 |
| Total catch | | | | |
| gr/dscf | 0.00251 | 0.00246 | 0.00134 | 0.00211 |
| gr/acf | 0.00227 | 0.00225 | 0.00125 | 0.00192 |
| lb/hr | 1.837 | 1.841 | 0.989 | 1.556 |
| Percent Isokinetic | 101.1 | 101.1 | 104.4 | |

## Table C-51. SUMMARY OF PARTICULATE TEST DATA -- CONVERTER SECONDARY HOOD OUTLET, PHELPS DODGE-AJO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/10/78 | 5/11/78 | 5/12/78 | |
| Test Duration - min. | 244 | 120 | 120 | 161 |
| Charge Rate - ton/hr | 27.3 | 10.6 | 28.4 | 22.1 |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 86,369 | 87,708 | 85,698 | 86,591 |
| Temperature (°F) | 143 | 163 | 162 | 156 |
| Stream (vol. %): | | | | |
| Water | 0.5 | 0.7 | 0.5 | 0.6 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.23 | 0.34 | 0.38 | 0.32 |
| Emissions - Particulate | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0756 | 0.0910 | 0.0793 | 0.0820 |
| gr/acf | 0.0688 | 0.0828 | 0.0835 | 0.0749 |
| lb/hr | 55.93 | 68.37 | 58.23 | 60.85 |
| Total catch | | | | |
| gr/dscf | 0.1016 | 0.1490 | 0.1105 | 0.1204 |
| gr/acf | 0.0925 | 0.1355 | 0.1025 | 0.1101 |
| lb/hr | 75.16 | 111.9 | 81.13 | 89.41 |
| Percent Isokinetic | 101.1 | 101.1 | 104.4 | |

## Table C-52. SUMMARY OF ARSENIC TEST DATA -- CONVERTER SECONDARY HOOD OUTLET, PHELPS DODGE-HIDALGO SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 7/25/78 | 7/26/78 | 7/26/78 | |
| Test Duration - min. | 300 | 240 | 240 | 260 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 69,076 | 83,346 | 56,063 | 69,495 |
| Temperature (°F) | 217 | 208 | 216 | 213 |
| Stream (vol. %): | | | | |
| Water | 1.5 | 2.2 | 2.2 | 2.0 |
| $CO_2$ | 0.2 | 0.2 | 0.2 | 0.2 |
| $O_2$ | 20.2 | 20.2 | 20.2 | 20.2 |
| $SO_2$ | 0.38 | 0.48 | 1.10 | 0.65 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00017 | 0.00010 | 0.00017 | 0.00014 |
| gr/acf | 0.00014 | 0.00008 | 0.00013 | 0.00012 |
| lb/hr | 0.1026 | 0.0718 | 0.0807 | 0.0850 |
| Total catch | | | | |
| gr/dscf | 0.00050 | 0.00011 | 0.00020 | 0.00026 |
| gr/acf | 0.00038 | 0.00009 | 0.00016 | 0.00021 |
| lb/hr | 0.2947 | 0.0778 | 0.0965 | 0.1563 |
| Percent Isokinetic | 85.3 | 98.6 | 99.1 | |

## Table C-53. SUMMARY OF ARSENIC TEST DATA -- CALCINE/ROASTER FUGITIVES BAGHOUSE INLET, PHELPS DODGE-DOUGLAS SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/3/78 | 5/4/78 | 5/4/78 | |
| Test Duration - min. | 55 | 49 | 38 | 48 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | 0.38 | 7.96 | 5.32 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 29,697 | 30,359 | 26,739 | 28,932 |
| Temperature (°F) | 76 | 72 | 65 | 71 |
| Stream (vol. %): | | | | |
| Water | 0.5 | 0.8 | 2.0 | 1.1 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.13 | 0.21 | 0.18 | 0.17 |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0000003 | 0.0000367 | 0.000193 | 0.000062 |
| gr/acf | 0.0000002 | 0.0000314 | 0.000155 | 0.000062 |
| lb/hr | 0.0001 | 0.0096 | 0.0424 | 0.0170 |
| Total catch | | | | |
| gr/dscf | 0.000026 | 0.000098 | 0.000281 | 0.000114 |
| gr/acf | 0.000023 | 0.000083 | 0.000234 | 0.000113 |
| lb/hr | 0.0067 | 0.0254 | 0.0645 | 0.0322 |
| Percent Isokinetic | 97.4 | 91.7 | 110.0 | |

Table C-54. SUMMARY OF ARSENIC TEST DATA -- CALCINE/ROASTER FUGITIVES BAGHOUSE OUTLET, PHELPS DODGE-DOUGLAS SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/3/78 | 5/4/78 | 5/4/78 | |
| Test Duration - min. | 65 | 42 | 40 | 49 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | 7.11 | a | 7.96 | 5.82 |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 31,539 | 32,296 | 31,781 | 31,872 |
| Temperature (°F) | 73 | 65 | 79 | 73 |
| Stream (vol. %): | | | | |
| Water | 1.0 | 0.9 | 1.4 | 1.1 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.08 | 0.19 | 0.15 | 0.14 |
| **Emissions - Arsenic** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.000011 | 0.000030 | 0.000069 | 0.000037 |
| gr/acf | 0.000010 | 0.000026 | 0.000058 | 0.000031 |
| lb/hr | 0.0030 | 0.0082 | 0.0188 | 0.0100 |
| Total catch | | | | |
| gr/dscf | 0.000105 | 0.000138 | 0.000095 | 0.000107 |
| gr/acf | 0.000090 | 0.000120 | 0.000079 | 0.000092 |
| lb/hr | 0.0283 | 0.0381 | 0.0259 | 0.0293 |
| Percent Isokinetic | 96.5 | 95.0 | 90.0 | |

Table C-55. SUMMARY OF PARTICULATE TEST DATA -- CALCINE/ROASTER FUGITIVES BAGHOUSE INLET, PHELPS DODGE-DOUGLAS SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 5/3/78 | 5/4/78 | 5/5/78 | |
| Test Duration - min. | 56 | 47 | 36 | 46 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| **Stack Effluent** | | | | |
| Flow rate (dscfm) | 30,294 | 29,153 | 29,036 | 29,380 |
| Temperature (°F) | 69 | 74 | 65 | 69 |
| Stream (vol. %): | | | | |
| Water | 1.2 | 0.0 | 0.6 | 0.6 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.13 | 0.21 | 0.18 | 0.17 |
| **Emissions - Particulate** | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 1.766 | 3.067 | 2.692 | 2.508 |
| gr/acf | 1.523 | 2.615 | 2.282 | 2.107 |
| lb/hr | 458.3 | 765.8 | 669.6 | 631.2 |
| Total catch | | | | |
| gr/dscf | 1.822 | 3.166 | 2.925 | 2.638 |
| gr/acf | 1.571 | 2.699 | 2.480 | 2.250 |
| lb/hr | 472.9 | 790.5 | 727.5 | 663.6 |
| Percent Isokinetic | 96.8 | 95.0 | 105.4 | |

## Table C-56. SUMMARY OF PARTICULATE TEST DATA -- CALCINE/ROASTER FUGITIVES BAGHOUSE OUTLET, PHELPS DODGE-DOUGLAS SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 5/3/78 | 5/4/78 | |
| Test Duration - min. | 65 | 42 | 54 |
| Charge Rate - ton/hr | 40.1 | 39.5 | 39.8 |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 29,018 | 30,985 | 30,002 |
| Temperature (°F) | 73 | 65 | 69 |
| Stream (vol. %): | | | |
| Water | 1.3 | 0.6 | 0.95 |
| $CO_2$ | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.09 | 0.19 | 0.14 |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 0.0031 | 0.0150 | 0.0091 |
| gr/acf | 0.0026 | 0.0131 | 0.0078 |
| lb/hr | 0.771 | 3.98 | 2.37 |
| Total catch | | | |
| gr/dscf | 0.0447 | 0.1927 | 0.1187 |
| gr/acf | 0.0387 | 0.1655 | 0.1021 |
| lb/hr | 11.11 | 51.16 | 31.79 |
| Percent Isokinetic | 93.6 | 87.7 | |

Table C-57. SUMMARY OF ARSENIC TEST DATA -- CONCENTRATE DRYER SCRUBBER OUTLET, KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 11/14/78 | 11/14/78 | 11/14/78 | |
| Test Duration - min. | 90 | 90 | 90 | 90 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 43,489 | 38,677 | 47,225 | 43,130 |
| Temperature (°F) | 166 | 129 | 118 | 138 |
| Stream (vol. %): | | | | |
| Water | 18.1 | 18.2 | 15.6 | 17.3 |
| $CO_2$ | a | a | a | a |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.06 | 0.07 | 0.07 | 0.07 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00003 | 0.00046 | 0.00111 | 0.00047 |
| gr/acf | 0.00002 | 0.00041 | 0.00084 | 0.00042 |
| lb/hr | 0.0120 | 0.1531 | 0.3739 | 0.1797 |
| Total catch | | | | |
| gr/dscf | 0.00003 | 0.00047 | 0.00099 | 0.00050 |
| gr/acf | 0.00003 | 0.00041 | 0.00090 | 0.00045 |
| lb/hr | 0.0143 | 0.1551 | 0.4029 | 0.1908 |
| Percent Isokinetic | 102.8 | 98.4 | 96.5 | |

## Table C-58. SUMMARY OF ARSENIC TEST DATA -- ACID PLANT INLET, KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 11/6/78 | 11/7/78 | 11/8/78 | |
| Test Duration - min. | 124 | 119 | 120 | 121 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 47,978 | 44,725 | 44,643 | 45,782 |
| Temperature (°F) | 421 | 476 | 409 | 436 |
| Stream (vol. %): | | | | |
| Water | 5.0 | 3.0 | 4.0 | 4.0 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 3.9 | 2.4 | 3.0 | 3.1 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.0034 | 0.0034 | 0.0020 | 0.0029 |
| gr/acf | 0.0029 | 0.0028 | 0.0018 | 0.0025 |
| lb/hr | 1.397 | 1.294 | 0.784 | 1.158 |
| Total catch | | | | |
| gr/dscf | 0.0055 | 0.0034 | 0.0021 | 0.0037 |
| gr/acf | 0.0047 | 0.0029 | 0.0018 | 0.0031 |
| lb/hr | 2.278 | 1.302 | 0.802 | 1.461 |
| Percent Isokinetic | 97.9 | 94.1 | 106.9 | |

## Table C-59. SUMMARY OF ARSENIC TEST DATA -- MATTE TAPPING DUCT, KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 11/1/78 | 11/2/78 | 11/3/78 | |
| Test Duration - min. | 70 | 60 | 66 | 65 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 48,968 | 48,645 | 43,868 | 47,162 |
| Temperature (°F) | 126 | 111 | 119 | 119 |
| Stream (vol. %): | | | | |
| Water | 1.4 | 1.0 | 0.0 | 0.8 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.09 | 0.10 | 0.12 | 0.10 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00030 | 0.00052 | 0.00115 | 0.00064 |
| gr/acf | 0.00026 | 0.00045 | 0.00099 | 0.00056 |
| lb/hr | 0.1255 | 0.2185 | 0.4341 | 0.2594 |
| Total catch | | | | |
| gr/dscf | 0.00036 | 0.00087 | 0.00136 | 0.00085 |
| gr/acf | 0.00030 | 0.00075 | 0.00117 | 0.00074 |
| lb/hr | 0.1505 | 0.3641 | 0.5128 | 0.3425 |
| Percent Isokinetic | 107.3 | 97.1 | 103.5 | |

## Table C-60. SUMMARY OF ARSENIC TEST DATA -- SLAG TAPPING DUCT, KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 11/1/78 | 11/2/78 | 11/3/78 | |
| Test Duration - min. | 60 | 120 | 120 | 100 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 43,308 | 40,541 | 38,914 | 40,921 |
| Temperature (°F) | 73 | 91 | 71 | 78 |
| Stream (vol. %): | | | | |
| Water | 0.3 | 0.9 | 1.0 | 0.7 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.003 | 0.008 | 0.004 | 0.005 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00030 | 0.00013 | 0.00005 | 0.00016 |
| gr/acf | 0.00026 | 0.00011 | 0.00004 | 0.00013 |
| lb/hr | 0.1119 | 0.0443 | 0.0172 | 0.0578 |
| Total catch | | | | |
| gr/dscf | 0.00036 | 0.00033 | 0.00006 | 0.00025 |
| gr/acf | 0.00030 | 0.00029 | 0.00005 | 0.00022 |
| lb/hr | 0.1327 | 0.1152 | 0.0217 | 0.0899 |
| Percent Isokinetic | 89.9 | 93.6 | 97.0 | |

Table C-61. SUMMARY OF ARSENIC TEST DATA -- CONVERTER FUGITIVES (FULL CYCLE), KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 11/6/78 | 11/8/78 | 11/9/78 | |
| Test Duration - min. | 188 | 181 | 182 | 184 |
| Charge Rate - ton/hr | a | a | a | a |
| Arsenic Rate - lb/hr | a | a | a | a |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 94,684 | 90,187 | 92,967 | 92,613 |
| Temperature (°F) | 105 | 103 | 61 | 90 |
| Stream (vol. %): | | | | |
| Water | 0.0 | 0.8 | 1.0 | 0.6 |
| $CO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| $O_2$ | 20.0 | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.09 | 0.14 | 0.33 | 0.19 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00028 | 0.00013 | 0.00044 | 0.00028 |
| gr/acf | 0.00024 | 0.00011 | 0.00039 | 0.00025 |
| lb/hr | 0.2262 | 0.1011 | 0.3517 | 0.2263 |
| Total catch | | | | |
| gr/dscf | 0.00034 | 0.00030 | 0.00056 | 0.00040 |
| gr/acf | 0.00029 | 0.00026 | 0.00050 | 0.00035 |
| lb/hr | 0.2762 | 0.2364 | 0.4469 | 0.3198 |
| Percent Isokinetic | 100.8 | 103.3 | 93.8 | |

## Table C-62. SUMMARY OF ARSENIC TEST DATA -- ROLLOUT CONVERTER FUGITIVES, KENNECOTT-MAGNA SMELTER

| Run No. | 1 | 2 | Average |
|---|---|---|---|
| Date | 11/6/78 | 11/8/78 | |
| Test Duration - min. | 88 | 65 | 77 |
| Charge Rate - ton/hr | a | a | a |
| Arsenic Rate - lb/hr | a | a | a |
| Stack Effluent | | | |
| Flow rate (dscfm) | 83,303 | 81,777 | 82,540 |
| Temperature (°F) | 100 | 106 | 103 |
| Stream (vol. %): | | | |
| Water | 1.0 | 1.0 | 1.0 |
| $CO_2$ | a | a | a |
| $O_2$ | 20.0 | 20.0 | 20.0 |
| $SO_2$ | 0.03 | 0.05 | 0.04 |
| Emissions - Particulate | | | |
| Probe, cyclone, and filter catch | | | |
| gr/dscf | 0.00019 | 0.00052 | 0.00035 |
| gr/acf | 0.00016 | 0.00044 | 0.00030 |
| lb/hr | 0.1342 | 0.3781 | 0.2561 |
| Total catch | | | |
| gr/dscf | 0.00019 | 0.00057 | 0.00035 |
| gr/acf | 0.00017 | 0.00048 | 0.00031 |
| lb/hr | 0.1376 | 0.3971 | 0.2674 |
| Percent Isokinetic | 103.2 | 99.8 | |

Table C-63. SUMMARY OF ARSENIC TEST DATA -- VENTURI SCRUBBER INLET, KENNECOTT-HAYDEN SMELTER

| Run No. | 1 | 2 | 3 | 4* | Average |
|---|---|---|---|---|---|
| Date | 12/10/76 | 12/11/76 | 12/13/76 | 12/13/76 | |
| Test Duration - min. | 110 | 135 | 85 | 75 | 101 |
| Charge Rate - ton/hr | 61 | 64 | 63 | 64.5 | 63.2 |
| Arsenic Rate - lb/hr | 1.88 | 1.63 | 1.65 | 1.23 | 1.60 |
| Stack Effluent | | | | | |
| Flow rate (dscfm) | 16,971 | 16,847 | 19,323 | 19,011 | 18,038 |
| Temperature (°F) | 636 | 623 | 615 | 621 | 624 |
| Stream (vol. %): | | | | | |
| Water | 26.2 | 22.4 | 32.3 | 27.5 | 27.1 |
| $CO_2$ | a | a | a | a | a |
| $O_2$ | 4.2 | 4.2 | 4.2 | 4.2 | 4.2 |
| $SO_2$ | 12.4 | 12.4 | 12.4 | 12.4 | 12.4 |
| Emissions - Arsenic | | | | | |
| Probe, cyclone, and filter catch | | | | | |
| gr/dscf | 0.0116 | 0.0108 | 0.0097 | 0.0072 | 0.0098 |
| gr/acf | 0.0044 | 0.0035 | 0.0036 | 0.0027 | 0.0036 |
| lb/hr | 1.69 | 1.56 | 1.60 | 1.17 | 1.50 |
| Total catch | | | | | |
| gr/dscf | 0.0129 | 0.0113 | 0.0100 | 0.0076 | 0.0104 |
| gr/acf | 0.0049 | 0.0045 | 0.0037 | 0.0029 | 0.0037 |
| lb/hr | 1.88 | 1.63 | 1.65 | 1.23 | 1.60 |
| Percent Isokinetic | 146 | 157 | 135 | 126 | |

*No corresponding test run was performed at the outlet.

## Table C-64. SUMMARY OF ARSENIC TEST DATA -- VENTURI SCRUBBER OUTLET, KENNECOTT-HAYDEN SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 12/10/76 | 12/11/76 | 12/13/76 | |
| Test Duration - min. | 145 | 145 | 140 | 143 |
| Charge Rate - ton/hr | 61 | 64 | 63 | 62.7 |
| Arsenic Rate - lb/hr | 0.04 | 0.02 | 0.02 | 0.03 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 15,493 | 18,918 | 18,017 | 17,476 |
| Temperature (°F) | 114 | 111 | 83 | 103 |
| Stream (vol. %): | | | | |
| Water | 9.8 | 9.1 | 3.8 | 7.6 |
| $CO_2$ | a | a | a | a |
| $O_2$ | 5.4 | 5.4 | 5.4 | 5.4 |
| $SO_2$ | 11.4 | 11.4 | 11.4 | 11.4 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00007 | 0.00006 | 0.00004 | 0.00006 |
| gr/acf | 0.00006 | 0.00005 | 0.00004 | 0.00006 |
| lb/hr | 0.010 | 0.010 | 0.006 | 0.007 |
| Total catch | | | | |
| gr/dscf | 0.00028 | 0.00012 | 0.00014 | 0.00018 |
| gr/acf | 0.00024 | 0.00010 | 0.00013 | 0.00016 |
| lb/hr | 0.037 | 0.019 | 0.022 | 0.027 |
| Percent Isokinetic | 112 | 115 | 101 | |

## Table C-65. SUMMARY OF ARSENIC TEST DATA --ACID PLANT OUTLET, KENNECOTT-HAYDEN SMELTER

| Run No. | 1 | 2 | 3 | Average |
|---|---|---|---|---|
| Date | 12/10/76 | 12/11/76 | 12/13/76 | |
| Test Duration - min. | 370 | 265 | 310 | 315 |
| Charge Rate - ton/hr | 61 | 64 | 63 | 62.7 |
| Arsenic Rate - lb/hr | 0.47 | 0.79 | 1.46 | 0.92 |
| Stack Effluent | | | | |
| Flow rate (dscfm) | 74,746 | 60,114 | 77,798 | 70,886 |
| Temperature (°F) | 155 | 158 | 175 | 163 |
| Stream (vol. %): | | | | |
| Water | 0.0 | 0.0 | 0.0 | 0.0 |
| $CO_2$ | a | a | a | a |
| $O_2$ | 7.7 | 7.7 | 7.7 | 7.7 |
| $SO_2$ | 0.0 | 0.0 | 0.0 | 0.0 |
| Emissions - Arsenic | | | | |
| Probe, cyclone, and filter catch | | | | |
| gr/dscf | 0.00014 | 0.00038 | 0.00020 | 0.00024 |
| gr/acf | 0.00012 | 0.00033 | 0.00016 | 0.00020 |
| lb/hr | 0.093 | 0.203 | 0.130 | 0.142 |
| Total catch | | | | |
| gr/dscf | 0.0007 | 0.0015 | 0.0022 | 0.0015 |
| gr/acf | 0.0006 | 0.0013 | 0.0018 | 0.0012 |
| lb/hr | 0.448 | 0.773 | 1.47 | 0.911 |
| Percent Isokinetic | 107 | 108 | 95 | |

Table C-66. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--ROASTER CALCINE DISCHARGE INTO LARRY CARS, ASARCO-TACOMA

| Run No. | Date | Observer 1 |  | Observer 2 |  | Mean duration of operation, min:sec | Mean % time emissions observed |
|---|---|---|---|---|---|---|---|
|  |  | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed |  |  |
| 1 | 6/24 | 1:20 | 0 | 1:15 | 0 | 1:18 | 0 |
| 2 | 6/24 | 2:40 | 0 | 2:40 | 0 | 2:40 | 0 |
| 3 | 6/24 | 1:20 | 0 | 1:20 | 0 | 1:20 | 0 |
| 4 | 6/25 | 1:23 | 0 | 1:23 | 0 | 1:23 | 0 |
| 5 | 6/25 | 1:58 | 0 | 1:52 | 0 | 1:55 | 0 |
| 6 | 6/25 | 1:42 | 0 | 1:42 | 0 | 1:42 | 0 |
| 7 | 6/25 | 1:12 | 0 | 1:13 | 0 | 1:13 | 0 |
| 8 | 6/25 | 1:20 | 0 | 1:20 | 0 | 1:20 | 0 |
| 9 | 6/26 | 2:50 | 0 | 2:49 | 0 | 2:50 | 0 |
| 10 | 6/26 | 1:48 | 0 | 1:48 | 0 | 1:48 | 0 |
| 11 | 6/26 | 2:30 | 0 |  |  | 2:30 | 0 |
| 12 | 6/26 | 1:42 | 0 |  |  | 1:42 | 0 |
| 13 | 6/26 | 3:04 | 0 |  |  | 3:04 | 0 |
|  |  |  |  |  | Average | 1:54 | 0 |

Table C-67. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--
MATTE TAP PORT AND MATTE LAUNDER, ASARCO-TACOMA

| Run[a] | Date | Observer 1 |  | Observer 2 |  | Mean |  |
|---|---|---|---|---|---|---|---|
|  |  | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed |
| 1 | 6/24 | 6:24 | 0 | 6:36 | 1 | 6:30 | 0.5 |
| 2 | 6/24 | 6:00 | 0 | 6:00 | 0 | 6:00 | 0 |
| 3 | 6/24 | 4:51 | 0 | 4:55 | 3 | 4:53 | 1.5 |
| 4 | 6/24 | 6:05 | 0 | 6:10 | 0 | 6:08 | 0 |
| 5[b] | 6/24 |  |  |  |  |  |  |
| 6 | 6/25 | 5:28 | 0 |  |  | 2:58 | 0 |
| 7 | 6/25 | 5:22 | 0 | 5:22 | 0 | 5:22 | 0 |
| 8 | 6/25 | 5:36 | 0 | 5:36 | 0 | 5:36 | 0 |
| 9 | 6/25 | 5:08 | 0 | 5:10 | 0 | 5:09 | 0 |
| 10 | 6/25 | 6:02 | 0 | 5:33 | 0 | 5:48 | 0 |
| 11 | 6/25 | 5:12 | 0 | 5:13 | 0 | 5:13 | 0 |
| 12 | 6/25 |  |  | 6:37 | 0 | 6:37 | 0 |
| 13 | 6/25 | 4:50 | 0 | 4:53 | 0 | 4:52 | 0 |
| 14 | 6/25 | 5:23 | 0 | 5:22 | 0 | 5:23 | 0 |
| 15 | 6/25 | 5:17 | 0 | 5:18 | 0 | 5:18 | 0 |
| 16[b] | 6/25 |  |  |  |  |  |  |
| 17 | 6/25 | 5:13 | 0 |  |  | 5:13 | 0 |
| 18 | 6/25 | 5:58 | 0 |  |  | 5:58 | 0 |
|  |  |  |  | Average |  | 5:26 | 0.13 |

[a] Method 22 data for corresponding runs at the matte discharge into the ladle are presented in Table C-68.

[b] Observations were made only at the matte discharge into the ladle; see Table C-68.

Table C-68. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--
MATTE DISCHARGE INTO LADLE, ASARCO-TACOMA

| Run[a] | Date | Observer 1 | | Observer 2 | | Mean | |
|---|---|---|---|---|---|---|---|
| | | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed |
| 1 | 6/24 | 6.30 | 0 | | | 6:30 | 0 |
| 2 | 6/24 | 5:49 | 0 | 5:40 | 0 | 5:45 | 0 |
| 3 | 6/24 | 4:53 | 0 | 5:01 | 0 | 4:57 | 0 |
| 4 | 6/24 | 6:12 | 0 | 6:10 | 0 | 6:11 | 0 |
| 5 | 6/24 | | | 6:31 | 0 | 6:31 | 0 |
| 6[b] | 6/25 | | | | | | |
| 7 | 6/25 | 5:09 | 0 | 5:02 | 0 | 5:06 | 0 |
| 8 | 6/25 | 5:21 | 0 | 5:28 | 0 | 5:25 | 0 |
| 9 | 6/25 | 5:02 | 0 | 5:03 | 0 | 5:03 | 0 |
| 10 | 6/25 | 4:29 | 0 | 4:32 | 0 | 4:31 | 0 |
| 11 | 6/25 | 5:12 | 0 | 5:13 | 0 | 5:13 | 0 |
| 12 | 6/25 | 6:16 | 0 | | | 6:16 | 0 |
| 13 | 6/25 | 4:43 | 0 | 4:45 | 0 | 4:44 | 0 |
| 14 | 6/25 | 5:13 | 0 | 5:15 | 0 | 5:14 | 0 |
| 15 | 6/25 | 5:15 | 0 | 5:09 | 0 | 5:12 | 0 |
| 16 | 6/25 | 5:41 | 0 | 5:50 | 0 | 5:46 | 0 |
| 17[b] | 6/25 | | | | | | |
| 18[b] | 6/25 | | | | | | |
| | | | | Average | | 5:30 | 0 |

[a] Method 22 data for corresponding runs at the matte tap and launder are presented in Table C-67.

[b] Observations were made only at the matte tap and launder; see Table C-67.

Table C-69. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--
SLAG TAP PORT AND SLAG LAUNDER, ASARCO-TACOMA

| Run[a] | Date | Observer 1 |  | Observer 2 |  | Mean Duration of operation, min:sec | Mean % Time emissions observed |
|---|---|---|---|---|---|---|---|
|  |  | Duration of operation, min:sec | % Time emissions observed | Duration of operation, min:sec | % Time emissions observed |  |  |
| 1 | 6/24 | 12:25[b] | 98[b] | 12:26[b] | 99[b] | 12:26 |  |
| 2 | 6/24 | 22:00 | 15 | 21:36 | 0 | 21:43 | 8 |
| 3 | 6/24 | 14:07 | 35[b] | 13:52[b] | 97[b] | 14:07 |  |
| 4[c] | 6/24 | 14:10 | 13 |  |  | 14:10 | 13 |
| 5 | 6/25 | 16:44 | 11 |  |  | 16:44 | 11 |
| 6[c] | 6/25 | 17:26 | 2 |  |  | 17:26 | 2 |
| 7 | 6/26 | 16:14 | 1 |  |  | 16:41 | 1 |
| 8 | 6/26 | 13:45 | 0.3 |  |  | 13:45 | 0 |
| 9 | 6/26 | 15:45 | 0 |  |  | 15:45 | 0 |
| 10 | 6/26 | 14:29 | 0 |  |  | 14:29 | 0 |
| 11[d] |  |  |  |  |  |  |  |
|  |  |  |  | Average |  | 15:40 | 4 |
|  |  |  |  | Std. dev. |  |  | 11 |

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[a] Method 22 data for corresponding runs at the slag skim discharge point appear in Table C-71.

[b] Observations were made at the entire slag tap process line including the slag tap port, slag launder, and slag discharge into ladle; and therefore not included in computing the mean of observations.

[c] Method 9 data for corresponding runs appear in Table C-70.

[d] No data obtained by Method 22.

Table C-70. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 9--
SLAG TAP AND SLAG LAUNDER, ASARCO-TACOMA

| Run | Date | Duration of operation, min. | Mean opacity, % | Maximum opacity, % |
|---|---|---|---|---|
| 1 | 6/25 | 14.75 | 1.3 | 10 |
| 2 | 6/25 | 18 | 10.3 | 30 |
| Average | | 16.38 | 6 | |
| Maximum | | | | 30 |

[a] Emission data were taken during entire slag tapping operation.

Table C-71. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--SLAG TAPPING AT SLAG DISCHARGE INTO POTS, ASARCO-TACOMA

| Run[a,b] | Date | Observer 1 | | Observer 2 | | Mean duration of operation, min:sec | Mean % time emissions observed |
|---|---|---|---|---|---|---|---|
| | | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed | | |
| 1 | 6/24 | 12:46 | 97 | 12:26 | 73 | 12:36 | 85 |
| 2 | 6/24 | 21:09 | 93 | 21:43 | 99 | 21:26 | 96 |
| 3 | 6/24 | 14:06 | 97 | 13:52 | 95 | 13:59 | 96 |
| 4 | 6/24 | 14:05 | 82 | | | 14:05 | 82 |
| 5 | 6/25 | 16:34 | 91 | | | 16:34 | 91 |
| 6 | 6/25 | 17:29 | 94 | | | 17:29 | 94 |
| 7 | 6/26 | 15:54 | 90 | | | 15:54 | 90 |
| 8 | 6/26 | 13:48 | 86 | | | 13:48 | 86 |
| 9 | 6/26 | 15:48 | 77 | | | 15:48 | 77 |
| 10 | 6/26 | 14:11 | 72 | | | 14:11 | 72 |
| 11 | 6/26 | 14:45 | 82 | | | 14:45 | 82 |
| | | | | | Average | 15:31 | 86 |
| | | | | | Std. dev. | | 8 |

[a] Visible emissions observation data by EPA Method 9 for corresponding runs are presented in Table C-72.

[b] Visible emissions observation data for corresponding runs for the slag tap port and launder are presented in Table C-69.

Table C-72. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 9--
SLAG TAPPING AT SLAG DISCHARGE INTO POTS, ASARCO-TACOMA

| Run[b] | Date | Duration of operation, min. | Mean opacity, % | Maximum opacity, % |
|---|---|---|---|---|
| 1 | 6/24 | c | | |
| 2 | 6/24 | c | | |
| 3 | 6/24 | c | | |
| 4 | 6/25 | 13.75 | 22.7 | 50 |
| 5 | 6/25 | 16.75 | 11.3 | 30 |
| 6 | 6/25 | 11.75[d] | 16 | 35 |
| 7 | 6/26 | 15 | 14.8 | 40 |
| 8 | 6/26 | 15 | 10.3 | 20 |
| 9 | 6/26 | 13 | 5.5 | 10 |
| 10 | 6/26 | 15 | 3.7 | 10 |
| 11 | 6/26 | | | |
| Average | | 14.32 | 12 | |
| Maximum | | | | 50 |

[a] Emission data were taken during entire slag tapping operation.

[b] Method 22 data for corresponding runs appear in Table C-71.

[c] No data were obtained by Method 9.

[d] Reading started after filling of first slag pot.

## Table C-73. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 22--CONVERTER SLAG RETURN TO REVERBERATORY FURNACE, ASARCO-TACOMA

| Run[a] | Date | Observer 1 |  | Observer 2 |  | Observer 3 |  |  |  |
|---|---|---|---|---|---|---|---|---|---|
|  |  | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed | Duration of operation, min:sec | % time emissions observed | Mean duration of operation, min:sec | Mean % time emissions observed |
| 1 | 6/24 | 1:04 | 100 | 1:05 | 89 | 0:58 | 100 | 1:04 | 96 |
| 2 | 6/24 | 0:47 | 97 | 0:47 | 96 | 0:46 | 100 | 0:46 | 98 |
| 3 | 6/24 | 0:54 | 100 | 0:53 | 100 | 0:55 | 100 | 0:53 | 100 |
| 4 | 6/25 | 0:55 | 100 |  |  |  |  | 0:55 | 100 |
| 5 | 6/25 |  |  | 1:03 | 100 |  |  | 1:03 | 100 |
| 6 | 6/25 |  |  |  |  | 0:52 | 100 | 0:52 | 100 |
| 7 | 6/25[b] |  |  |  |  |  |  |  |  |
| 8 | 6/26 | 1:04 | 66 |  |  |  |  | 1:04 | 66 |
| 9 | 6/26 | 1:00 | 85 |  |  |  |  | 1:00 | 85 |
| 10 | 6/26 | 1:15 | 83 |  |  |  |  | 1:15 | 83 |
| 11 | 6/26 | 0:55 | 82 | 0:41 | 93 |  |  | 0:48 | 88 |
| 12 | 6/26 |  |  |  |  |  |  |  |  |
|  |  |  |  |  |  | Average |  | 0:58 | 92 |
|  |  |  |  |  |  | Std. dev. |  |  | 11 |

[a] Visible emissions observation data by EPA Method 9 for corresponding runs are presented in Table C-74.

[b] No data obtained by Method 22.

## Table C-74. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 9--CONVERTER SLAG RETURN TO REVERBERATORY FURNACE, ASARCO-TACOMA

| Run | Date | Observer 1 | | | Observer 2 | | |
|---|---|---|---|---|---|---|---|
| | | Duration of operation, min. | Average opacity, % | Maximum opacity, % | Duration of operation, min. | Average opacity, % | Maximum opacity, % |
| 1 | 6/24 | a | | | | | |
| 2 | 6/24 | a | | | | | |
| 3 | 6/24 | a | | | | | |
| 4 | 6/25 | 1.00 | 17.5 | 30 | 1.00 | 16 | 25 |
| 5 | 6/25 | 1.25 | 20 | 40 | | | |
| 6 | 6/25 | | | | 1.00 | 23 | 35 |
| 7 | 6/25 | 0.75 | 23 | 35 | 0.75 | 23 | 30 |
| 8 | 6/26 | 1.25 | 5 | 10 | | | |
| 9 | 6/26 | 1.25 | 11 | 20 | | | |
| 10 | 6/26 | 1.50 | 12 | 20 | | | |
| 11 | 6/26 | 1.25 | 13 | 20 | | | |
| 12 | 6/26 | 0.75 | 5 | 10 | | | |

average opacity for all readings - 15%
maximum opacity during all readings - 40%

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[a] Data were not obtained by Method 9 on 6/24/80.

## Table C-75. VISIBLE EMISSIONS OBSERVATION DATA, EPA METHOD 9--BLISTER DISCHARGE FROM CONVERTER AT THE TAMANO SMELTER IN JAPAN[a,b,c]

| Opacity, % | Total time equal to or greater than given opacity | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | 1st blister discharge | | 2nd blister discharge | | 3rd blister discharge | | Total blister discharge[d] | |
| | min: sec. | % of total time | min: sec. | % of total time | min: sec. | % of total time | min. sec: | % of total time |
| 5 | 8:00 | 53 | 11:30 | 96 | 1:00 | 29 | 20:30 | 67 |
| 10 | 5:00 | 33 | 8:45 | 73 | 0:30 | 14 | 14:15 | 47 |
| 15 | 3:15 | 22 | 5:15 | 44 | | | 8:30 | 28 |
| 20 | 1:30 | 10 | 3:15 | 27 | | | 4:45 | 16 |
| 25 | 0:30 | 3 | 2:00 | 17 | 0.15 | 7 | 2:45 | 8 |
| 30 | 0:15 | 2 | 0.45 | 6 | | | 1:00 | 3 |
| 35 | | | 0.15 | 2 | | | 0:15 | < 1 |

[a] Observation point: converter secondary hood system.

[b] Data were based on a total of 30 5-minute observations for three successive blister discharges of the total four blister discharges during one converter cycle. Duration of each of the three discharges observed were 15 minutes, 12 minutes, and 3.5 minutes, respectively.

[c] Table C-76 summarizes the observation data into average opacities for each set of 6-minute data.

[d] Total of the three individual blister discharges.

Table C-76. SUMMARY OF AVERAGE OBSERVED OPACITIES FOR BLISTER DISCHARGE AT THE TAMANO SMELTER IN JAPAN[a]

| Set No.[b] | Average Opacity[c],% |
|---|---|
| 1 | 6 |
| 2 | 8 |
| 3 | 11 |
| 4 | 10 |
| 5 | 9 |

[a] Based on same observation data used for Table C-75.

[b] Observation time for each set was 6 minutes.

[c] Average of all sets is 9 percent.

### Table C-77. SUMMARY OF EPA METHOD 9 VISIBLE EMISSIONS DATA--INDIVIDUAL AND TOTAL MATTE CHARGES TO CONVERTER OBSERVED AT THE TAMANO SMELTER IN JAPAN[a,b,c,d]

| | Total time equal to or greater than given opacity | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | 1st matte charge | | 2nd matte charge | | 3rd matte charge | | 4th matte charge | | Total matte charge | |
| Opacity, % | min: sec. | % of total time | min: sec. | % of total time | min: sec. | % of total time | min: sec. | % of total time | min: sec. | % of total time |
| 5 | 0:45 | 43 | 0:45 | 60 | 0:45 | 43 | 0 | | 2:15 | 35 |
| 10 | | | 0:15 | 20 | 0:15 | 14 | | | 0:30 | 8 |
| 25 | 0:15 | 14 | | | | | | | 0:15 | 4 |

a) Matte charges 1, 2 and 3 were successive charges; respective charging times for matte charges 1, 2, 3 and 4 were 1.75 min., 1.25 min., 1.75 min., and 1.75 min.

b) Observation point: converter secondary hood system.

c) Data were based on a total of 6.5 minute observation for three successive matte charges at the beginning of one converter cycle and an intermediate matte charging during the cycle. Average duration of each matte charge was 1.5 minutes.

d) Total of the four individual matte charges; average opacity for matte charging, based on total observation, is 3.0 percent.

Table C-78. SUMMARY OF VISIBLE EMISSIONS OBSERVATION DATA--COPPER BLOW AT THE TAMANO SMELTER IN JAPAN[a]

| Set No. [b] | Average Opacity, % |
|---|---|
| 1 | 0 |
| 2 | 0 |
| 3 | 0 |
| 4 | 0 |

[a] Observation point: converter secondary hood system.

[b] Each set is based on 6-minute observation.

Table C-79. SUMMARY OF VISIBLE EMISSIONS OBSERVATION DATA--SLAG BLOW AT THE TAMANO SMELTER IN JAPAN[a]

| Set No.[b] | Average Opacity, % |
|---|---|
| 1 | 0 |
| 2 | 0 |
| 3 | 0 |
| 4 | 0 |
| 5 | 0 |

[a] Observation point: converter secondary hood system.

[b] Each set represents a 6-minute observation. Set Nos. 1 and 2 are based on the 1st slag blow and set nos. 3 through 5 are based on the second slag blow of the three slag blow total of the complete converter cycle.

### Table C-80. SUMMARY OF VISIBLE EMISSIONS OBSERVATION DATA--CONVERTER SLAG DISCHARGE AT THE TAMANO SMELTER IN JAPAN[a]

| Set No.[b] | Average Opacity, % |
|---|---|
| 1 | 0 |
| 2 | 0 |

[a] Observation point: converter secondary hood system.

[b] Each of two consecutive sets of 6-minute observations are made during one slag discharge.

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# APPENDIX D

# TEST METHODS

# TEST METHODS

## D.1 EMISSION MEASUREMENT METHODS

At the beginning of the testing program, a literature search was conducted to identify available sampling and analytical techniques for determining arsenic emissions. The search revealed that most arsenic emissions are in the form of arsenic trioxide and arsenic pentoxide. According to the literature, the most commonly used arsenic sampling method has been filtration; however, a number of reports have indicated that filtration alone is not adequate, even at ambient temperatures, because arsenic trioxide is a potentially volatile material. Since it was decided to determine the amount of arsenic collected as a particulate, the Method 5 train, with back-up impinger collectors, was chosen as the starting point for the arsenic sampling system. Based on the available information, a dilute sodium hydroxide solution was chosen as a collecting solution for the impingers. This, however, presented a problem since many of the gas streams to be sampled had very high concentrations of sulfur dioxide ($SO_2$), some as high as 3.5 percent. Therefore, a series of impingers containing hydrogen peroxide was placed between the filter and the first impinger containing sodium hydroxide to remove the $SO_2$. This was the configuration for the "working train" used during the first four field tests.

Analytical methods for arsenic were better defined in the literature. The most commonly-used procedure is a wet chemical method based on arsine generation, but certain metals including copper are interfering agents with this method. Instrumental techniques include atomic absorption, neutron activation, and x-ray fluorescence. Atomic absorption spectrophotometry (AAS) was chosen as the most promising technique because of its ready availability, familiarity, and low cost; however, arsenic absorbs weakly and only in the extreme ultra-violet area of the spectrum (193.7 nm). At that wavelength, molecular absorption by flame gases and solution species can interfere with arsenic detection. Despite this, conventional AAS can still be used, provided that: (1) the fuel and combustion gas are carefully chosen and nonatomic background correction is used; and (2) arsenic concentrations are relatively high. However, for lower arsenic concentrations, the interference effects necessitate the use of a special, more sensitive technique, such as the hydride generator or the carbon rod (flameless) system. Before testing began, both conventional and special AAS methods were compared and evaluated, in terms of their accuracy, precision, and sensitivity.

During the first two field tests, samples were collected with the working train and analyzed either by conventional or carbon rod AAS depending on the arsenic concentration. The analytical results showed that 95 to 100 percent of the arsenic was collected ahead of the NaOH impingers. In the course of analyzing these samples, the following detailed sample preparation procedure was developed. Solid samples were digested with 0.1 N sodium hydroxide, extracted with concentrated nitric acid, evaporated to dryness, and then redissolved in dilute nitric acid. Liquid samples were treated similarly except that there was no need for the sodium hydroxide digestion step. Advantages of the sample preparation procedure include: (1) reduction of the level of the collected sulfuric acid in the liquid sample fraction; (2) dissolution of the arsenic in the solid samples; and (3) production of a similar solution matrix for all the different sample fractions.

After the second test, questions were raised about the sampling and analytical procedures. First of all, laboratory studies of vaporized arsenic trioxide showed no difference in the arsenic collection efficiency of 0.1 N sodium hydroxide and pure water. These results indicated that the arsenic collection mechanism is condensation and that any condenser would be an effective collector. Consequently, the conventional Method 5 train (with $H_2O$ impingers) was suggested as an alternate to the working train and simultaneous testing of the two trains was planned for the next facility.

Second, an evaluation of the different AAS techniques for low-concentration uncovered some precision and accuracy problems with the carbon rod method when large quantities of dissolved solids (particularly sulfates) are present. The hydride generator technique, it was found, gives much more precise and accurate results in the presence of dissolved solids. In view of this, it was decided that all future low-concentration arsenic samples (i.e., too low for conventional AA analysis) would be analyzed by the hydride generator method.

Third, concern was expressed that arsenic was being lost in the evaporation step of sample preparation. To investigate this, recovery studies were performed on standard samples. These studies showed that there is no significant loss of arsenic during the evaporation step.

Fourth, additional studies showed that while arsenic trioxide is soluble in alkaline, acid, and neutral solutions, its rate of dissolution is slow except in alkaline solutions. Therefore, the clean-up procedure for future test was modified, to require that the train be rinsed with 0.1 N sodium hydroxide to insure removal of condensed arsenic.

Fifth, a comparison of arsenic extraction techniques indicated that higher arsenic yields (by up to 200 percent) can be obtained from smelter particulate when a method capable of dissolving the entire sample is used instead of the less rigorous acid extraction procedure. As a result, it was decided that in future tests, filters would be analyzed by both methods, until more conclusive filter extraction data were obtained.

During the third and fourth field tests, the working train was used for sampling, but additional runs were taken during the fifth test using paired trains of the working and alternate procedures. Analysis of the samples from the paired tests showed no significant difference in collection efficiency. Therefore, the final recommendation was to use the alternate train, since it is easier to operate and analyze. During the fifth and final field test, the alternate train was used.

Filters from the third, fourth, and fifth field tests were extracted, using both the total dissolution and acid extraction procedure. The results showed that filters extracted by the less rigorous method could in some cases yield 25 percent less arsenic than if totally dissolved. Based upon these results, the final recommendation was to extract the filters first by the simple acid extraction; then, if any undissolved sample remained, to extract the undissolved solids by the total dissolution method.

## D.2 CONTINUOUS MONITORING

There is currently no available method for continuously monitoring arsenic emissions. For purposes of demonstrating proper operation and maintenance of control devices, continuous monitors are available for measuring opacity from baghouses or electrostatic precipitators, and measuring pressure drop across scrubbers. However, these measurements are not necessarily indicators of the magnitude of arsenic emissions and should not be used for compliance determinations. In addition, opacity may not be applicable as an indicator of proper operation and maintenance where baghouses and precipitators are used to control captured fugitive emissions because of the uncontrolled particulate is very low in concentration.

The recommended monitoring program for continually assessing arsenic emissions is a periodic application of the performance test Method 108 as recommended in Part D-3 below. This is the only method evaluated at this time for demonstration of compliance with arsenic emissions.

## D.3 PERFORMANCE TEST METHODS

The recommended performance test method for arsenic is Method 108. Based on the development work already discussed, the method uses the Method 5 train for sampling, 0.1 N sodium hydroxide for cleanup, and either conventional or hydride generator AAS for sample analysis. In order to perform Method 108, Methods 1 through 4 must also be used. Subpart A or 40 CFR 60 requires that facilities subject to standards of performance for new stationary sources be constructed so as to provide sampling ports adequate for the applicable test methods, and platforms, access, and utilities necessary to perform testing at those ports.

Sampling costs for performing a test consisting of three Method 108 runs is estimated to range from $10,000 to $14,000. If in-plant personnel are used to conduct tests, the costs will be somewhat less.

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# APPENDIX E

# QUANTITATIVE EXPRESSIONS OF PUBLIC CANCER RISKS FROM EMISSIONS OF INORGANIC ARSENIC FROM HIGH-ARSENIC PRIMARY COPPER SMELTERS

# QUANTITATIVE EXPRESSIONS OF PUBLIC CANCER RISKS FROM THE EMISSIONS OF INORGANIC ARSENIC FROM HIGH-ARSENIC PRIMARY COPPER SMELTERS

## E.1 INTRODUCTION

### E.1.1 Overview

The quantitative expressions of public cancer risks presented in this appendix are based on (1) a dose-response model that numerically relates the degree of exposure to airborne inorganic arsenic to the risk of getting lung cancer, and (2) numerical expressions of public exposure to ambient air concentrations of inorganic arsenic estimated to be caused by emissions from stationary sources. Each of these factors is discussed briefly below and details are provided in the following sections of this appendix.

### E.1.2 The Relationship of Exposure to Cancer Risk

The relationship of exposure to the risk of getting lung cancer is derived from epidemiological studies in occupational settings rather than from studies of excess cancer incidence among the public. The epidemiological methods that have successfully revealed associations between occupational exposure and cancer for substances such as asbestos, benzene, vinyl chloride, and ionizing radiation, as well as for inorganic arsenic, are not readily applied to the public sector, with its increased number of confounding variables, much more diverse and mobile exposed population, lack of consolidated medical records, and almost total absence of historical exposure data. Given such uncertainties, EPA considers it improbable that any association, short of very large increases in cancer, can be verified in the general population with any reasonable certainty by an epidemiological study. Furthermore, as noted by the National Academy of Sciences (NAS)[1], "...when there is exposure to a material, we are not starting at an origin of zero cancers. Nor are we starting at an origin of zero carcinogenic agents in our environment. Thus, it is likely that any carcinogenic agent added to the environment will act by a particular mechanism on a particular cell population that is already being acted on by the same mechanism to induce cancers." In discussing experimental dose-response curves, the NAS observed that most information on carcinogenesis is derived from studies of ionizing radiation with experimental animals and with humans which indicate a linear no-threshold dose-response relationship at low doses. They added that although some evidence exists for thresholds in some animal tissues, by and large, thresholds have not been established for most tissues. NAS concluded that establishing such low-dose thresholds "...would require massive, expensive, and impractical experiments ..." and recognized that the U.S. population "...is a large, diverse, and genetically heterogeneous group exposed to a large variety of toxic agents." This fact, coupled with the known genetic variability to carcinogenesis and the predisposition of some individuals to some form of cancer, makes it extremely difficult, if not impossible, to identify a threshold.

For these reasons, EPA has taken the position, shared by other Federal regulatory agencies, that in the absence of sound scientific evidence to the contrary, carcinogens should be considered to pose some cancer risk at any exposure level. This no-threshold presumption is based on the view that as little as one molecule of a carcinogenic substance may be sufficient to transform a normal cell into a cancer cell. Evidence is available from both the human and animal health literature that cancers may arise from a single transformed cell. Mutation research with ionizing radiation in cell cultures indicates that such a transformation can occur as the result of interaction with as little as a single cluster of ion pairs. In reviewing the available data regarding carcinogenicity, EPA found no compelling scientific reason to abandon the no-threshold presumption for inorganic arsenic.

In developing the exposure-risk relationship for inorganic arsenic, EPA has assumed that a linear no-threshold relationship exists at and below the levels of exposure reported in the epidemiological studies of occupational exposure. This means that any exposure to inorganic arsenic is assumed to pose some risk of lung cancer and that the linear relationship between cancer risks and levels of public exposure is the same as that between cancer risks and levels of occupational exposure. EPA believes that this assumption is reasonable for public health protection in light of presently available information. However, it should be recognized that the case for the linear no-threshold dose-response relationship model for inorganic arsenic is not quite as strong as that for carcinogens which interact directly or in metabolic form with DNA. Nevertheless, there is no adequate basis for dismissing the linear no-threshold model for inorganic arsenic. The exposure-risk relationship used by EPA represents only a plausible upper-limit risk estimate in the sense that the risk is probably not higher than the calculated level and could be much lower.

The numerical constant that defines the exposure-risk relationship used by EPA in its analysis of carcinogens is called the unit risk estimate. The unit risk estimate for an air pollutant is defined as the lifetime cancer risk occurring in a hypothetical population in which all individuals are exposed continuously from birth throughout their lifetimes (about 70 years) to a concentration of one $\mu g/m^3$ of the agent in the air which they breathe. Unit risk estimates are used for two purposes: (1) to compare the carcinogenic potency of several agents with each other, and (2) to give a crude indication of the public health risk which might be associated with estimated air exposure to these agents. The comparative potency of different agents is more reliable when the comparison is based on studies of like populations and on the same route of exposure, preferably inhalation.

The unit risk estimate for inorganic arsenic that is used in this appendix was prepared by combining the three different exposure-risk numerical constants developed from three occupational studies.[2] The unit risk estimate is expressed as a range that reflects the statistical uncertainty associated with combining the three exposure-risk relationships. The methodology used to develop the unit risk estimate is described in E.2 below. EPA is updating its health effects assessment document for inorganic arsenic. A preliminary estimate by EPA's health scientists is that the unit risk estimate may change.

### E.1.3 Public Exposure

The unit risk estimate is only one of the factors needed to produce quantitative expressions of public health risks. Another factor needed is a numerical expression of public exposure, i.e., of the numbers of people exposed to the various concentrations of inorganic arsenic. The difficulty of defining public exposure was noted by the National Task Force on Environmental Cancer and Health and Lung Disease in their 5th Annual Report to Congress, in 1982.[3] They reported that "...a large proportion of the American population works some distance away from their homes and experience different types of pollution in their homes, on the way to and from work, and in the workplace. Also, the American population is quite mobile, and many people move every few years." They also noted the necessity and difficulty of dealing with long-term exposures because of "...the long latent period required for the development and expression of neoplasia [cancer]..."

EPA's numerical expression of public exposure is based on two estimates. The first is an estimate of the magnitude and location of long-term average ambient air concentrations of inorganic arsenic in the vicinity of emitting sources based on dispersion modeling using long-term estimates of source emissions and meteorological conditions. The second is an estimate of the number and distribution of people living in the vicinity of emitting sources based on Bureau of Census data which "locates" people by population centroids in census tract areas. The people and concentrations are combined to produce numerical expressions of public exposure by an approximating technique contained in a computerized model. The methodology is described in E.3 below.

### E.1.4 Public Cancer Risks

By combining numerical expressions of public exposure with the unit risk estimate, two types of numerical expressions of public cancer risks are produced. The first, called individual risk, relates to the person or persons estimated to live in the area of highest concentration as estimated by the dispersion model. Individual risk is expressed as "maximum lifetime risk." As used here, the word "maximum" does not mean the greatest possible risk of cancer to the public. It is based only on the maximum exposure estimated by the procedure used. The second, called aggregate risk, is a summation of all the risks to people estimated to be living within the vicinity (usually within 20 kilometers) of a source and is customarily summed for all the sources in a particular category. The aggregate risk is expressed as incidences of cancer among all of the exposed population after 70 years of exposure; for statistical convenience, it is often divided by 70 and expressed as cancer incidences per year. These calculations are described in more detail in E.4 below.

There are also risks of nonfatal cancer and of serious genetic effects, depending on which organs receive the exposure. No numerical expressions of such risks have been developed; however, EPA considers all of these risks when making regulatory decisions on limiting emissions of inorganic arsenic.

## E.2 THE UNIT RISK ESTIMATE FOR INORGANIC ARSENIC[2]

### E.2.1 The Linear No-Threshold Model for Estimation of Unit Risk Based on Human Data (General)[4]

Very little information exists that can be utilized to extrapolate from high exposure occupational studies to low environmental levels. However, if a number of simplifying assumptions are made, it is possible to construct a crude dose-response model whose parameters can be estimated using vital statistics, epidemiologic studies, and estimates of worker exposures. In human studies, the response is measured in terms of the relative risk of the exposed cohort of individuals compared to the control group. The mathematical model employed assumes that for low exposures the lifetime probability of death from lung cancer (or any cancer), P, may be represented by the linear equation

$$P = A + B_H x \qquad (1)$$

where A is the lifetime probability of cancer in the absence of the agent, x is the average lifetime exposure to environmental levels in micrograms per cubic meter ($\mu g/m^3$) and $B_H$ is the increased probability of cancer associated with each $\mu g/m^3$ increase of the agent in air.

If we make the assumption that R, the relative risk of lung cancer for exposed workers, compared to the general population, is independent of the length or age of exposure but depends only upon the average lifetime exposure, it follows that

$$R = \frac{P}{P_0} = \frac{A + B_H\ (x_0 + x_1)}{A + B_H\ (x_0)} \tag{2}$$

or

$$RP_0 = A + B_H\ (x_0 + x_1) \tag{3}$$

where $x_0$ = lifetime average exposure to the agent for the general population, $x_1$ = lifetime average exposure to the agent in the occupational setting, and $P_0$ = lifetime probability of respiratory cancer applicable with no or negligible arsenic exposure. Substituting $P_0 = A + B_H\ x_0$ and rearranging gives

$$B_H = P_0\ (R - 1)/x_1 \tag{4}$$

To use this model, estimates of R and $x_1$ must be obtained from the epidemiologic studies. The value $P_0$ is derived from the age-cause-specific death rates for combined males found in 1976 U.S. Vital Statistics tables using the life table methodology. For lung cancer the estimate of $P_0$ is 0.036.[4]

### E.2.2 The Unit Risk Estimate for Inorganic Arsenic[2]

As noted in the health effects assessment document[5] for inorganic arsenic, there are numerous occupational studies which relate increased cancer rates to arsenic exposure. Based on these studies, it is concluded in the health assessment document that there is substantial evidence that inorganic arsenic is a human carcinogen. However, many of these studies are inappropriate for use in developing a unit risk estimate for inorganic arsenic because the route of exposure was not by inhalation or because it was impossible to make a reasonable estimate of the population's lifetime average exposure.

Three studies, Lee and Fraumeni (1969), Ott et al. (1974), and Pinto et al. (1977), contained enough pertinent information to make independent quantitative estimates of human cancer risks due to human exposures to atmospheric arsenic. The crudeness of the exposure estimates in those studies is due to such factors as high variability in the chemical measurement of arsenic, a scarcity of monitoring data, and the necessity of working from summarized data tables presented in the literature rather than complete data on all individuals. However, by accepting the data in spite of its recognized limitations, and making a number of simplifying assumptions concerning dose-response relationships and exposure patterns, it was possible to estimate the carcinogenic potency of arsenic. Using a linear model, it was estimated that the increase in the lung cancer rate per increase of 1 $\mu g/m^3$ of atmospheric arsenic was 9.4% (Pinto et al.), 17.0% (Ott et al.), and 3.3% (Lee and Fraumeni). The consistency of these estimates is very good considering the relative crudeness of the data upon which they are based. The geometric mean of the rate estimates from the three studies was calculated to be 8.1%. Using this value as a best estimate and applying equation 4, one calculates the unit risk estimate of $2.95 \times 10^{-3}$ per $\mu g/m^3$.

If we assume that the linear model and exposure estimates are correct, so that the only source of uncertainty is from combining results from the three different studies, a 95% confidence interval for the above unit risk estimate may be obtained. Upper and lower 95% confidence limits can be obtained by multiplying the unit risk estimate by about 4 and 0.25 respectively. Thus, the 95% statistical confidence limits for the unit risk estimate range from $7.5 \times 10^{-4}$ to $1.2 \times 10^{-2}$.

## E.3 QUANTITATIVE EXPRESSIONS OF PUBLIC EXPOSURE TO INORGANIC ARSENIC EMITTED FROM HIGH-ARSENIC PRIMARY COPPER SMELTERS

### E.3.1 EPA's Human Exposure Model (HEM) (General)

EPA's Human Exposure Model is a general model capable of producing quantitative expressions of public exposure to ambient air concentrations of pollutants emitted from stationary sources. HEM contains (1) an atmospheric dispersion model, with included meteorological data, and (2) a population distribution estimate based on Bureau of Census data. The only input data needed to operate this model are source data, e.g., plant location, height of the emission release point, and temperature of the off-gases. Based on the source data, the model estimates the magnitude and distribution of ambient air concentrations of the pollutant in the vicinity of the source. The model is programmed to estimate these concentrations within a radial distance of 20 kilometers from the source. If other radial distances are preferred, an over-ride feature allows the user to select the distance desired. The selection of 20 kilometers as the programmed distance is based on modeling considerations, not on health effects criteria or EPA policy. The dispersion model contained in HEM is felt to be reasonably accurate within 20 kilometers. If the user wishes to use a dispersion model other than the one contained in HEM to estimate ambient air concentrations in the vicinity of a source, HEM can accept the concentrations if they are put into an appropriate format. (Note that for high-arsenic copper smelters, the Industrial Source Complex Dispersion Model was used to estimate ambient air concentrations. This dispersion model is described in section E.3.2 and the exposure methodology is described in section E.3.3.)

Based on the radial distance specified, HEM combines numerically the distributions of pollutant concentrations and people to produce quantitative expressions of public exposure to the pollutant.

### E.3.2 Pollutant Concentrations Near a Source

The dispersion analysis uses the Industrial Source Complex Model Long-Term (ISCLT). Since more information was available for dispersion modeling of high-arsenic primary copper smelters than for the other arsenic-emitting source categories, the ISCLT model was used in place of the dispersion model contained in HEM. The ISCLT model is an EPA approved, validated model (validated for $SO_2$) that is more sophisticated than the HEM dispersion model. The ISCLT output was put in a format acceptable to the exposure portion of HEM. The ISCLT model is applicable in areas of flat to gently rolling terrain free from channeling and thermal effects associated with large bodies of water. In this analysis, the model was also exercised in the "rural" mode, which allowed for consideration of moderately stable atmospheric conditions. Model options for elevated terrain and building downwash were also used. Building downwash refers to trapped air in eddies on the opposite side of a building from the emission source. For the low-level emissions sources at this smelter, use of the building downwash option significantly improves the accuracy of the model estimates at receptors lying close to the source. Nevertheless, the model appears to consistently underestimate concentrations close in. Of the comparisons made, approximately half of the model estimates lie within a factor of two of the observed concentrations.[6]

This dispersion analysis used meteorological data collected from the Puget Sound Air Pollution Control Agency's tower at North 26th and Pearl Streets in Tacoma, Washington (approximately 10 kilometers south of the smelter) and concurrent surface observations at McChord Air Force Base approximately 14.5 kilometers south of the smelter) during the year 1972.

These data were compiled into a stability wind rose. Ambient air temperatures and mixing heights as a function of stability class and wind speed data, from the report Assessment of the Air Quality Impact of $S_2$ Emissions from the ASARCo-Tacoma Smelter, were also used in the analysis./

The following criteria were used to select the radial distances at which concentrations were computed:

- The shortest distance, 0.3 kilometers, was based on the location of expected high concentrations from ground-level sources asociated with the smelter.
- The next two distances, 0.8 and 1.5 kilometers, were based on the location of the highest expected concentrations from the two different sets of stack parameters for the converter source. The distances computed were ranked on the basis of the computed concentration times the frequency of occurrence for a particular combination of wind speed and stability class.
- The remaining distances were selected on the basis of the highest expected concentrations from the main stack acting alone. Terrain height and wind direction were the primary criteria in making this determination. A profile of terrain points was established consisting of the highest elevations at various distances from the plant, primarily in the northeastern and southwestern quadrants, the directions of predominant wind flow. The concentrations computed by the ISC model using the actual meteorological data were then ranked to select the terrain points and thus the corresponding distances from the smelter.

All terrain points and receptor elevations are taken from U.S. Geological Survey maps, at a scale of 1:24,000. Modeling considerations, however, disallow receptor elevations above the elevation of any source. As a result, the sources are broken into three groups (main stack and two ground sources) that make use of two receptor elevation data sets. For the main stack the elevations are limited to approximately 70 to 216 meters (200 to 710 feet) main stack level (MSL). The groundlevel sources designated G(1) and G(2) in Figure E-1 are modeled assuming flat terrain. The total estimated concentrations from the smelters are then computed as the sum of these two individual source contributions.

The main stack was treated as a point source. Due to its relatively low exit velocity, a stack-tip downwash adjustment to plume height is made, as necessary. The roof opening, Source G(2), is treated as three separate emissions points subject to building downwash. The building width is designated as the diameter of a circular structure of the same area as the actual building. Source G(1), representing numerous ground sources, is composed of two emission areas, 93 meters on a side, and a third emission area, 185 meters on a side, which together roughly correspond to the triangle depicted in Figure E-1.

### E.3.3 The People Living Near A Source

To estimate the number and distribution of people residing within 20 kilometers of a plant, the model contains a slightly modified version of the "Master Enumeration District List--Extended" (MED-X) data base. The data base is broken down into enumeration district/block group (ED/BG) values. MED-X contains the population centroid coordinates (latitude and longitude) and the 1970 population of each ED/BG in the United States (50 States plus the District of Columbia). For human exposure estimates, MED-X

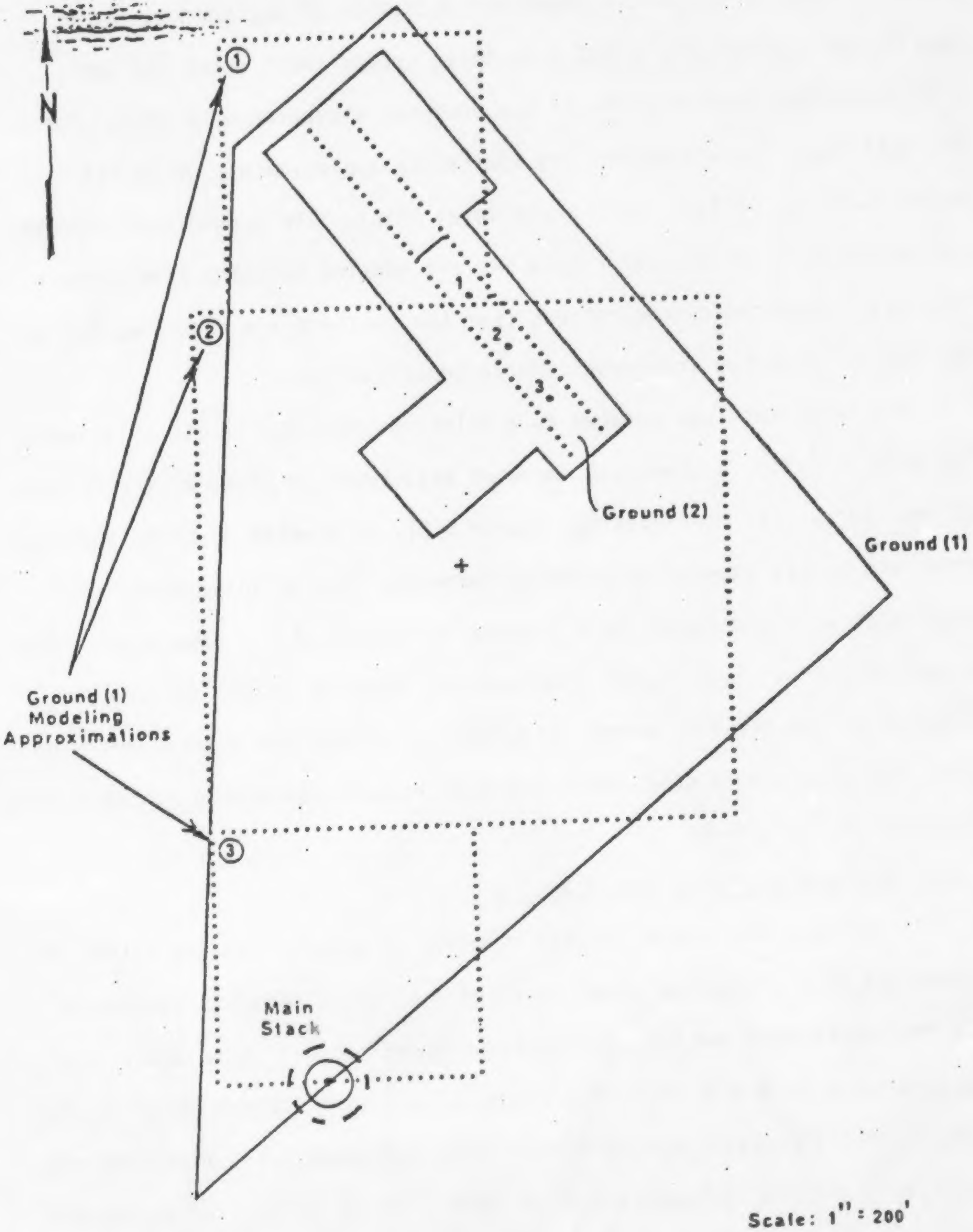
N
①
②
③
1
2
3
Ground (2)
Ground (1)
Ground (1)
Modeling
Approximations
Main
Stack
Scale: 1" = 200'

Figure E-1. Tacoma Plant Configuration

has been reduced from its complete form (including descriptive and summary data) to produce a computer file of the data necessary for the estimation. A separate file of county-level growth factors, based on 1978 estimates of the 1970 to 1980 growth factor at the county level, has been used to estimate the 1980 population for each ED/BG. HEM identifies the population around each plant by using the geographical coordinates of the plant. The HEM identifies, selects, and stores for later use those ED/BGs with coordinates falling within 20 kilometers of plant center.

### E.3.4 Exposure[8]

The Human Exposure Model (HEM) uses the estimated ground level concentrations of a pollutant, in this case from ISCLT, together with population data to calculate public exposure. For each of 144 receptors located around a plant, the concentration of the pollutant and the number of people estimated by the HEM to be exposed to that particular concentration are identified. The HEM multiplies these two numbers to produce exposure estimates and sums these products for each plant.

A two-level scheme has been adopted in order to pair concentrations and populations prior to the computation of exposure. The two level approach is used because the concentrations are defined on a radius-azimuth (polar) grid pattern with non-uniform spacing. At small radii, the grid cells are usually smaller than ED/BG's; at large radii, the grid cells are usually larger than ED/BG's. The area surrounding the source is divided into two regions, and each ED/BG is classified by the region in which its centroid lies. Population exposure is calculated differently for the ED/BG's located within each region. For ED/BG centroids located between 0.3 km and 2.8 km from the emission source, populations are divided between neighboring concentration grid points. There are 64 (4 x 16) polar grid points within this range. Each grid point has a polar sector defined by two concentric arcs and two wind direction radials. Each of these grid points and respective concentrations are assigned to the nearest ED/BG centroid identified from MED-X. Each ED/BG can be paired with one or many concentration points. The population associated with the ED/BG centroid is then divided among all concentration grid points assigned to it. The land area within each polar sector is considered in the apportionment.

For population centroids between 2.8 km and 18.8 km from the source, a concentration grid cell, the area approximating a rectangular shape bounded by four receptors, is much larger than the area of a typical ED/BG. Since there is an approximate linear relationship between the logarithm of concentration and the logarithm of distance for receptors more than 2 km from the source, the entire population of the ED/BG is assumed to be exposed to the concentration that is logarithmically interpolated radially and arithmetically interpolated azimuthally from the four receptors bounding the grid cell. Concentration estimates for 80 (5 x 16) grid cell receptors at 4.4, 8.5, 11.5, 15.0, and 18.8 km from the source along each of 16 wind directions are used as reference points for this interpolation.

In summary, two approaches are used to arrive at coincident concentration/population data points. For the 64 concentration points within 2.8 km of the source, the pairing occurs at the polar grid points using an apportionment of ED/BG population by land area. For the remaining portions of the grid, pairing occurs at the ED/BG centroids themselves through the use of log-log and linear interpolation. (For a more detailed discussion of the model used to estimate exposure, see Reference 6 and 7.)

### E.3.5 Public Exposure to Inorganic Arsenic Emissions from High Arsenic Primary Copper Smelters

#### E.3.5.1 Source Data

One smelter is included in the analysis. Table E.1 lists the name and address of the plant considered, and Table E.2 lists the plant data used as input to the Human Exposure Model (HEM).

#### E.3.5.2 Exposure Data

Table E.3 lists, the total number of people encompassed by the exposure analysis and the total exposure. Total exposure is the sum of the products of number of people times the ambient air concentration to which they are exposed, as calculated by HEM. Table E.4 sums, for the plant, the numbers of people exposed to various ambient concentrations, as calculated by HEM.

## TABLE E-1

## IDENTIFICATION OF HIGH-ARSENIC PRIMARY COPPER SMELTERS

| Plant Number Code | Plant Name and Address |
|---|---|
| 1 | ASARCO, Inc.<br>TACOMA, WA |

Table E-2. Input Data to Dispersion Model for ASARCO-Tacoma Smelter[a] (Baseline Control)

| Emission Source | Height (m) | Diameter (m) | Temperature (°K) | Velocity (m/sec) | Emissions (kg/hr) |
|---|---|---|---|---|---|
| Stack | 61.0 | 7.3 | 366 | 9.7 | 17.3 |
| Ground (1) | 30.5 | b | 310 | 0.5 | 14.0 |
| Ground (2) | Ground | c | 294 | d | 1.3 |

a) Smelter location: Latitude 47° 17' 49"

Longitude 122° 30' 23"

b) Building roof monitor (see Figure E-1)

c) Three area sources (see Figure E-1)

d) Discharge velocity assumed to be negligible.

## TABLE E-3 TOTAL EXPOSURE AND NUMBER OF PEOPLE EXPOSED (HIGH-ARSENIC PRIMARY COPPER SMELTERS)*

| Plant | Total Number of People Exposed | Total Exposure (People - $\mu g/m^3$) |
|---|---|---|
| 1 | 368,000 | 103,000 |

---

* An 18.8-kilometer radius was used for the analysis of high-arsenic primary copper smelters.

Table E-4

## PUBLIC EXPOSURE FOR HIGH ARSENIC COPPER SMELTERS AS PRODUCED BY THE HUMAN EXPOSURE MODEL

| Concentration Level ($\mu g/m^3$) | Population Exposed (Persons)* | Exposure (Persons - $\mu g/m^3$)** |
|---|---|---|
| 30.6 | 9 | 290 |
| 25.0 | 9 | 290 |
| 10.0 | 223 | 3,240 |
| 2.5 | 1,316 | 10,200 |
| 1.0 | 4,455 | 20,000 |
| 0.5 | 16,997 | 38,800 |
| 0.25 | 36,051 | 52,000 |
| 0.10 | 84,005 | 69,300 |
| 0.05 | 253,783 | 95,100 |
| 0.025 | 352,848 | 103,071 |
| 0.01 | 368,409 | 104,520 |
| 0.005 | 368,409 | 104,680 |
| 0.0025 | 368,409 | 104,780 |
| 0.001 | 368,409 | 104,820 |
| 0.0005 | 368,409 | 104,850 |
| 0.0001 | 368,409 | 104,850 |
| $1.0 \times 10^{-9}$ | 368,409 | 104,850 |

*Column 2 displays the computed value, rounded to the nearest whole number, of the cumulative number of people exposed to the matching and higher concentration levels found in column 1. For example, 0.5 people would be rounded to 0 and 0.51 people would be rounded to 1.

**Column 3 displays the computed value of the cumulative exposure to the matching and higher concentation levels found in column 1.

## E.4 QUANTITATIVE EXPRESSIONS OF PUBLIC CANCER RISKS FROM INORGANIC ARSENIC EMITTED FROM HIGH-ARSENIC PRIMARY COPPER SMELTERS

### E.4.1 Methodology (General)

### E.4.1.1 The Two Basic Types of Risk

Two basic types of risk are dealt with in the analysis. "Aggregate risk" applies to all of the people encompassed by the particular analysis. Aggregate risk can be related to a single source, to all of the sources in a source category, or to all of the source categories analyzed. Aggregate risk is expressed as incidences of cancer among all of the people included in the analysis, after 70 years of exposure. For statistical convenience, it is often divided by 70 and expressed as cancer incidences per year. "Individual risk" applies to the person or persons estimated to live in the area of the highest ambient air concentrations and it applies to the single source associated with this estimate as estimated by the dispersion model. Individual risk is expressed as "maximum lifetime risk" and reflects the probability of getting cancer if one were continuously exposed to the estimated maximum ambient air concentration for 70 years.

### E.4.1.2 The Calculation of Aggregate Risk

Aggregate risk is calculated by multiplying the total exposure produced by HEM (for a single source, a category of sources, or all categories of sources) by the unit risk estimate. The product is cancer incidences among the included population after 70 years of exposure. The total exposure, as calculated by HEM, is illustrated by the following equation:

$$\text{Total Exposure} = \sum_{i=1}^{N} (P_i C_i)$$

where

$\sum$ = summation over all grid points where exposure is calculated

$P_i$ = population associated with grid point i,

$C_i$ = long-term average inorganic arsenic concentration at grid point i,

N = number of grid points to 2.8 kilometers and number of ED/BG centroids between 2.8 and 20 kilometers of each source.

To more clearly represent the concept of calculating aggregate risk, a simplified example illustrating the concept follows:

EXAMPLE

This example uses assumptions rather than actual data and uses only three levels of exposure rather than the large number produced by HEM. The assumed unit risk estimate is $3 \times 10^{-3}$ at 1 $\mu g/m^3$, and the assumed exposures are:

| ambient air concentrations | number of people exposed to given concentration |
|---|---|
| 2 $\mu g/m^3$ | 1,000 |
| 1 $\mu g/m^3$ | 10,000 |
| 0.5 $\mu g/m^3$ | 100,000 |

The probability of getting cancer if continuously exposed to the assumed concentrations for 70 years is given by:

| concentration | | unit risk | | probability of cancer |
|---|---|---|---|---|
| 2 $\mu g/m^3$ | x | $3 \times 10^{-3} (\mu g/m^3)^{-1}$ | = | $6 \times 10^{-3}$ |
| 1 $\mu g/m^3$ | x | $3 \times 10^{-3}$ " | = | $3 \times 10^{-3}$ |
| 0.5 $\mu g/m^3$ | x | $3 \times 10^{-3}$ " | = | $1.5 \times 10^{-3}$ |

The 70 year cancer incidence among the people exposed to these concentrations is given by:

| probability of cancer at each exposure level | | number of people at each exposure level | | cancer incidences after 70 years of exposure |
|---|---|---|---|---|
| $6 \times 10^{-3}$ | x | 1,000 | = | 6 |
| $3 \times 10^{-3}$ | x | 10,000 | = | 30 |
| $1.5 \times 10^{-3}$ | x | 100,000 | = | 150 |
| | | | TOTAL = | 186 |

The aggregate risk, or total cancer incidence, is 186 and, expressed as cancer incidence per year, is 186 ÷ 70, or 2.7 cancers per year. The total cancer incidence and cancers per year apply to the total of 111,000 people assumed to be exposed to the given concentrations.

### E.4.1.3 The Calculation of Individual Risk

Individual risk, expressed as "maximum lifetime risk," is calculated by multiplying the highest concentration to which the public is exposed, as reported by HEM, by the unit risk estimate. The product, a probability of getting cancer, applies to the number of people which HEM reports as being exposed to the highest listed concentration. The concept involved is a simple proportioning from the 1 $\mu g/m^3$ on which the unit risk estimate is based to the highest listed concentration. In other words:

$$\frac{\text{maximum lifetime risk}}{\text{highest concentration to which people are exposed}} = \frac{\text{the unit risk estimate}}{1\ \mu g/m^3}$$

### E.4.2 Risks Calculated for Emissions of Inorganic Arsenic from High-Arsenic Primary Copper Smelters

The explained methodologies for calculating maximum lifetime risk and cancer incidences were applied to the high-arsenic primary copper smelter, assuming a baseline level of emissions. A baseline level of emissions means the level of emissions after the application of controls either currently in place or required to be in place to comply with curent state or Federal regulations but before application of controls that would be required by a NESHAP.

Table E-5 summarizes the calculated risks. To understand the relevance of these numbers, one should refer to the analytical uncertainties discussed in section E.5 below.

### Table E-5 MAXIMUM LIFETIME RISK AND CANCER INCIDENCE FOR HIGH-ARSENIC PRIMARY COPPER SMELTERS
(Assuming Baseline Controls)

| Plant | Maximum Lifetime Risk | Cancer Incidences Per Year | Cancer Incidence (one case in [x] years) |
|---|---|---|---|
| 1 | $2.3 \times 10^{-2}$ - $3.6 \times 10^{-1}$ | 1.1 - 17.4 | 1 in 0.9 yr. - 1 in. 0.06 yr. |

TOTALS FOR THIS SOURCE CATGEGORY

| Number of Plants | Total Number of People Exposed (within 18.8 km) | Highest Individual Risk | Cancer Incidences | |
|---|---|---|---|---|
| | | | per year | one case in [x] years |
| 1 | 368,000 | $2.3 \times 10^{-2}$ - $3.6 \times 10^{-1}$ | 1.1 - 17.4 | 1 in 0.9 yr. - 1 in 0.06 yr. |

NOTE: The ranges in these quantifications of public health impacts reflect the uncertainty of combining the three different dose-response relationships relevant to the three occupational studies which EPA used as the basis for the development of unit risk estimate.[2]

## E.5 ANALYTICAL UNCERTAINTIES APPLICABLE TO THE CALCULATIONS OF PUBLIC HEALTH RISKS CONTAINED IN THIS APPENDIX

### E.5.1 The Unit Risk Estimate

The procedure used to develop the unit risk estimate is described in reference 2. The model used and its application to epidemiological data have been the subjects of substantial comment by health scientists. The uncertainties are too complex to be summarized sensibly in this appendix. Readers who wish to go beyond the information presented in the reference should see the following Federal Register notices: (1) OSHA's "Supplemental Statement of Reasons for the Final Rule", 48 FR 1864 (January 14, 1983); and (2) EPA's "Water Quality Documents Availability" 45 FR 79318 (November 28, 1980).

The unit risk estimate used in this analysis applies only to lung cancer. Other health effects are possible; these include skin cancer, hyperkeratosis, peripheral neuropathy, growth retardation and brain dysfunction among children, and increase in adverse birth outcomes. No numerical expressions of risks relevant to these health effects is included in this analysis.

### E.5.2 Public Exposure

#### E.5.2.1 General

The basic assumptions implicit in the methodology are that all exposure occurs at people's residences, that people stay at the same location for 70 years, that the ambient air concentrations and the emissions which cause these concentrations persist for 70 years, and that the concentrations are the same inside and outside the residences. From this it can be seen that public exposure is based on a hypothetical rather than a realistic premise.

It is not known whether this results in an over-estimation or an under-estimation of public exposure.

### E.5.2.2 The Public

The following are relevant to the public as dealt with in this analysis:

1. Studies show that all people are not equally susceptible to cancer. There is no numerical recognition of the "most susceptible" subset of the population exposed.

2. Studies indicate that whether or not exposure to a particular carcinogen results in cancer may be affected by the person's exposure to other substances. The public's exposure to other substances is not numerically considered.

3. Some members of the public included in this analysis are likely to be exposed to inorganic arsenic in the air in the workplace, and workplace air concentrations of a pollutant are customarily much higher than the concentrations found in the ambient, or public air. Workplace exposures are not numerically approximated.

4. Studies show that there is normally a long latent period between exposure and the onset of lung cancer. This has not been numerically recognized.

5. The people dealt with in the analysis are not located by actual residences. As explained previously, they are "located" in the Bureau of Census data for 1970 by population centroids of census districts. Further, the locations of these centroids has not been changed to reflect the 1980 census. The effect is that the actual locations of residences with respect to the estimated ambient air concentrations is not known and that the relative locations used in the exposure model have changed since the 1970 census.

6. Many people dealt with in this analysis are subject to exposure to ambient air concentrations of inorganic arsenic where they travel and shop (as in downtown areas and suburban shopping centers), where they congregate (as in public parks, sports stadiums, and schoolyards), and where they work outside (as mailmen, milkmen, and construction workers). These types of exposures are not numerically dealt with.

### E.5.2.3. The Ambient Air Concentrations

The following are relevant to the estimated ambient air concentrations of inorganic arsenic used in this analysis:

1. Flat terrain was assumed in the dispersion model. Concentrations much higher than those estimated would result if emissions impact on elevated terrain or tall buildings near a plant.

2. The increase in concentrations that could result from re-entrainment of arsenic-bearing dust from, e.g., city streets, dirt roads, and vacant lots, is not considered.

3. With few exceptions, the arsenic emission rates are based on assumptions rather than on emission tests. See the BID for details on each source.

## E.6 References

1. National Academy of Sciences, "Arsenic," Committee on Medical and Biological Effects of Environmental Pollutants, Washington, D.C., 1977. Docket Number (OAQPS 79-8) II-A-3.

2. The Carcinogen Assessment Group's Final Risk Assessment on Arsenic. OAQPS Docket Number 79-8-II-A-7. May 2, 1980.

3. U.S. EPA, et.al., "Environmental Cancer and Heart and Lung Disease," Fifth Annual Report to Congress by the Task Force on Environmental Cancer and Health and Lung Disease, August, 1982.

4. U.S. EPA, "Health Assessment Document for Acrylonitrile," Draft Report from the Office of Health and Assessment, EPA-600/8-82-007, November, 1982.

5. U.S. EPA, "An Assessment of the Health Effects of Arsenic," April 1978, Docket Number (OAQPS 79-8) II-A-5.

6. U.S. Environmental Protection Agency. An Evaluation Study of the Industrial Source Complex (ISC) Dispersion Model. U.S. Environmental Protection Agency. Research Triangle Park, North Carolina. EPA-450/4-81-002. 1981

7. U.S. Environmental Protection Agency. Assessment of the Air Quality Impact of $SO_2$ Emissions from the ASARCo-Tacoma Smelter. U.S. Environmental Protection Agency. Research Triangle Park, North Carolina. EPA-910/9-76-028. 1976

8. Systems Application, Inc., "Human Exposure to Atmospheric Concentrations of Selected Chemicals." (Prepared for the U.S. Environmental Protection Agency, Research Triangle Park, North Carolina). Volume I, Publication Number EPA-2/250-1, and Volume II, Publication Number EPA-1/250-2.

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15. SUPPLEMENTARY NOTES

16. ABSTRACT

Standards of performance to control emissions of inorganic arsenic from new and existing primary copper smelters processing feed materials containing an annual average of 0.7 percent or greater arsenic are being proposed under Section 112 of the Clean Air Act. This document provides information on the background and authority, regulatory alternatives considered, and environmental and economic impacts of the regulatory alternatives.

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